New Ways to Save Time and Money

---

at Mine, Mill and Smelter

New Ways to Save Time and Money

at Mine, Mill and Smelter

Practical operating ideas originated by and for men concerned with problems met in mining and preparing ores and minerals

COPYRIGHT, 1927

McGRAW-HILL PUBLISHING CO., INC.,
TENTH AVENUE AT 36TH STREET,
NEW YORK, N. Y.

Foreword

Regular subscribers to Engineering and Mining Journal obtain from the weekly issues of the paper short, helpful articles which contain many suggestions for saving time and money.

This book consists of a representative group of such articles. Practically all of them were contributed by active superintendents, engineers, metallurgists and operating officials.

The collection enables new subscribers active at mine, mill or smelter to profit by material of proved value to those seeking "how" and "why" suggestions of proved value.

CONTENTS

I—At the Mine—*Underground and on the Surface*

Page

Aligning Raises 59
Aligning Raises for Connections.................... 33
Car Locking Device for Shaft Cages.................. 38
Cementing Side Openings in Drill Holes.............. 85
Churn Drilling Fissured Ground...................... 76
Cost Sheet for the Mine Foreman..................... 73
Cushioning an Ore Feeder Arm........................57
Deepening the Pilares Shaft of Moctezuma Copper Company 63
Drilling and Blasting in United Verde's Open Pit Mine 90
Dump Doors in Winze Sinking......................... 83
Hot-water Heater 31
How to Clean the Oil Reservoir of a Gasoline Engine.. 62
Hydraulic Eductor for Sump Drainage................. 8
Junction of Inclined with Vertical Shaft............ 78
Maintaining Shaft-Plumbing Bob in Position.......... 36
Method of Shafting Alignment........................ 77
Mine Dewatering Diagram............................. 14
New Timber Preservative............................. 18
Notes on Tempering Drill Steel...................... 44
Overcoming Faulty Operation of Compressor Loader.. 54
Portable Poles for Electrified Open-pit Haulage...... 40
Recent Powder Magazine Construction................. 86
Reclaiming a Stockpile by Chinaman Chutes........... 61
Recovering Churn-Drill Tools........................ 56
Rehabilitating an Air Compressor.................... 9
Rope Guide for Idler Sheaves........................ 58
Scraper Loaders, as Designed and Used at Casapalca, Peru 20
Solenoid Lock for Incline Conveyors................. 18
Standard Hose Fittings at the United Verde.......... 101
Straightening and Bending Devices................... 74
Surveying Steep Inclined Shafts without an Auxiliary Telescope 68
Timber Frame for Underhand Square Setting........... 42
Timber Mats Aid Track Laying for Electric Shovel... 7
Trolley Wire Guard.................................. 60

(Continued on next page)

Contents *(Continued)*

Page

Trucks and Small Electric Shovels at United Verde... 13
Underground Ore-Bin Construction at the United Verde 70
Ventilating Stopes and Winzes...................... 5
Waste Ore and Transfer Gate...................... 34

II—In Mill, Smelter and Refinery

An Adjustable Drag Type Classifier................ 105
Ball, Roller and Plain Bearings................... 111
Cananea Smelter Improves Reverberatory Practice.... 129
Cleaning Device for Elevator Bucket................ 146
Compressed-Air Shovel 124
Concentration of Solution Values in Evaporating Cyanide Pulp Samples 125
Conveying Ore at the Ajo Plant.................... 156
Cyanide Precipitation Refinements.................. 152
Determination of Xanthate in Impure Solutions....... 148
Determination of Zinc in Roasted Ore................ 145
Electrical Timing Device for Ore Samplers.......... 127
Flow of Water through Narrow Weirs.............. 144
Handy Electric Light for Stamp Batteries............ 116
Keeping a Solution Tank from Freezing............. 113
Lime Circuit at Magna Mill........................ 123
Lock for Inclined Conveyors...................... 114
Metallurgical Control at the Magna Mill............. 109
Ninety-ton Lead-Remelting Furnace of Improved Heat Efficiency 121
Obviating the Harmful Effect of Soluble Salts in Flotation 141
Precipitation and Refining at Tonopah Extension Mill. 158
Pulp-Density Indicator and Recorder................ 131
Rapid Determination of Barium.................... 155
Recording Alkalinity or Acidity of Flotation Pulp..... 103
Repair of Roll Fly Wheel Pulley.................. 115
Repulping Box for Handling Filter Cakes............ 138
Roll Shell Heating Device........................ 139
Sampling of Concentrates in Bins................... 118
Scraper for Beaded Pan Conveyor................. 126
Scraper for Conveyor Pulley...................... 117
Specific Gravity Determination for Solids............ 133
Specific Gravity Indicator in Use at Dome Mines, Ltd. 107
Treatment of Roaster Flue Dust................... 134
Use of Portable Protecting Screens................. 145

New Ways to Save Time and Money

—at Mine, Mill and Smelter

I—At the Mine–
Underground and on the Surface

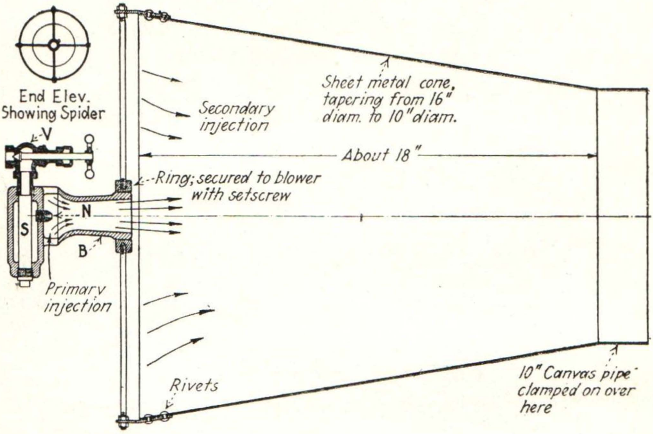
Ventilating Stopes and Winzes

By F. A. McLean

Ventilation of stopes, winzes, small prospects, shafts, or other workings not cleared by the ventilation system is sometimes a problem, as is also the case where it is impracticable to depend on the air currents set up by the exhaust from drills or to install rotary fans. In many mines it is customary to open air lines and use air at full line pressure to blow out smoke. This obviously wastes air and sometimes reduces pressure to a point where drills in other workings are handicapped.

A better and more efficient way of using compressed air for ventilation has been developed in several Australian mines where flat back stopes are in use. A small induction blower such as is used on rivet forges and steel-treating furnaces is the important feature. The induction blower has been on the market for three or four years and is intended to reduce the energy waste which occurs when forges are operated directly from high-pressure air lines. The cut on next page shows a sectional view of the blower.

B is the blast tube with pipe thread for attaching to the forge or other device; *N* is the high-pressure jet nozzle with an orifice of proper size and shape; and *S* is a strainer which insures freedom from interruption by particles of dirt. The strainer is cleaned, without


End Elev.
Showing Spider
V
Secondary injection
Sheet metal cone, tapering from 16" diam. to 10" diam.
About 18"
Ring; secured to blower with setscrew
N
S
B
Primary injection
Rivets
10" Canvas pipe clamped on over here

Sectional view of induction blower used for secondary ventilation

removal, by taking out the bottom plug and blowing air through it. *V* is the needle valve which regulates the air jet. In operation the compressed air issues from the nozzle at high velocity, and the energy due to its velocity is imparted to the surrounding air, which passes forward into the throat of the blast tube and is constantly replaced by additional free air flowing in at the bell-mouth openings of the blast tube. The efficiency of the device as given by manufacturer's rating is about 95 per cent, which signifies that 95 per cent of the air is induced and only 5 per cent (free air equivalent) is taken from the compressed-air line. Some idea of the air volume is given in the accompanying table, which represents tests made upon a small forge.

Result of Blower Tests

Cubic feet (free air) required	No. 1	No. 2	No. 3
Without blower	36.4	52.4	59.3
With blower	1.35	2.52	4.76
Draft in ash pit (inches of water)	0.35	0.70	1.35

A small rivet-heating forge requires an average of 45 cu.ft. of free air per minute. Directly taken from the compressed-air lines, this represents an expenditure of 7.5 hp. With the blower attachment, the consumption of air from the lines is less than 2 cu.ft. per minute (representing one-third of one horsepower) and the balance, or 43 cu.ft., per minute, is induced.

As used in Australia the blower is mounted in a spider so that the air from the blast tube is directed into the large end of a galvanized iron cone, as shown in the accompanying figure. The cone is 15 in. in diameter at the large end, about 18 in. long, and tapers to a diameter of 10 in. at the small end. A piece of 10-in. canvas tubing, with hoops at regular intervals to prevent collapse, is fitted to the small end of the cone and extends to the point where ventilation is required. The opening in the jet nozzle of the blower is enlarged from $\frac{1}{16}$ in., its normal diameter, to $\frac{1}{8}$ in., allowing a large volume of high-pressure air to pass through the blower, thus increasing the amount of induced air. Regulation of the amount of air delivered is obtained by adjusting the needle valve. The arrangement has worked well with lengths of canvas tubing up to 40 ft. It is not expected to replace other methods of ventilation, but is suggested as an auxiliary to the regular equipment. The distance at which ventilation could be effected would depend largely on local conditions.

Timber Mats Aid Track Laying for Electric Shovel

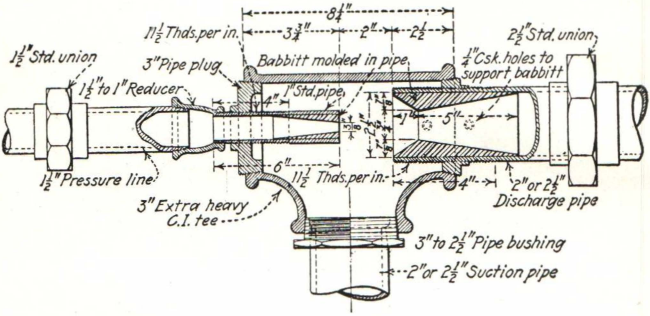
By Oliver O. Ormond

Superintendent, Wabigon Mine, Buhl, Minn.

Track laying ahead of the 350-ton electric shovel in the Hanna Ore Mining Co.'s Wabigon open pit, on the Mesabi range, is facilitated by the use of timber mats 8x14 ft. in dimensions to which corresponding lengths of track consisting of 130-lb. rails have been spiked. There are ten of these mats to a shovel, and they take the place of the longer sections of track formerly used. Four or five pit men were needed to carry the sections forward from behind the shovel. The mats require only two pit men. They are picked up by the shovel, swung around to the front, pulled back into position by means of a cable, and connected up by the men.

The mat consists of three layers of timber bolted together thus: The bottom layer has twelve 8 x 8 in. x 14 ft. timbers tied or bolted together with four $1\frac{1}{8}$-in. rods.

On this are laid twenty-one 6 x 8 in. x 8 ft. timbers placed at right angles to the timbers underneath. This second layer is bolted together with three rods. The two layers are then bolted to each other with about eighteen ¾x14-in. bolts with countersunk heads. Planking is then spiked to the upper layer of 6x8-in. timber. Sixteen 3 x 11 in. by 8 ft. planks plus a 4-in. strip are required. The planking is laid parallel with the layer immediately below it. A 1¼-in. round U-bolt is set in each corner of the mat to permit it to be handled by the shovel.



Hydraulic Eductor for Sump Drainage

A simple type of water eductor operated by water under pressure has been used at the Bingham and Pittsburgh-Idaho mines of the U. S. Smelting, Refining & Mining Co. The eductor is used for draining sumps, pockets, and shallow shafts while they are being deepened. The accompanying sketch shows the general design of the eductor, which was constructed in the shops. Suction is continuous and the eductor starts quickly. It is not particularly efficient, as about an equal quantity of pressure water as compared to the water removed from the excavation is required. However, it is a light, portable rig that does not take up much room. Its handiness overcomes its lack of efficiency. At an altitude of about 7,000 ft. the eductor will lift approximately 22 ft., or about 20 per cent of the available pressure head.

Rehabilitating an Air Compressor

By D. Welge
Consulting Electrical Engineer, Hobart Building,
San Francisco

Repairs to and the general conditioning of a mine air-compressor installation with a view to increase of efficiency and the elimination of various difficulties presented an interesting problem the details of which are of interest to master mechanics and others having to do with air compressors.

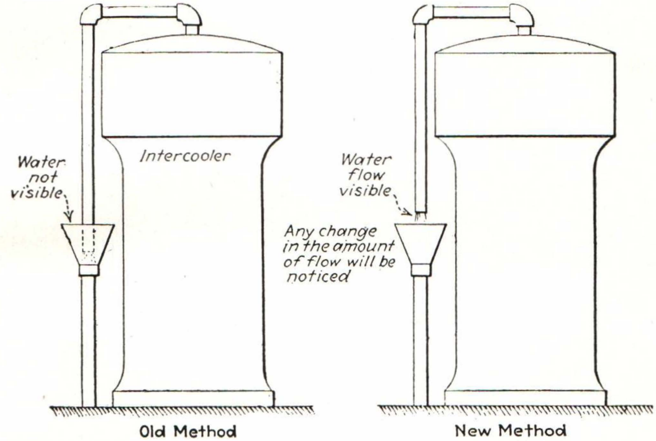


Fig. 1—Pipe arrangement for cooling water discharge

A thorough overhauling of the compressor was out of the question, as it had to be maintained in operation, and spare parts such as inlet valves, piston rings, and the like were not available. The operating condition of the compressor was good enough, but many little details required attention. First of these was cleanliness. A place was provided for all tools, packing, separate parts, and other small items. A special case was provided to hold new waste, avoiding the careless practice of throwing it into a corner of the compressor room. The compressor, which was of an old type, was built without oil gutters to catch the drips or oil thrown from the bearings. Sacks had been placed upon the floor to catch the drip. These were removed and drip

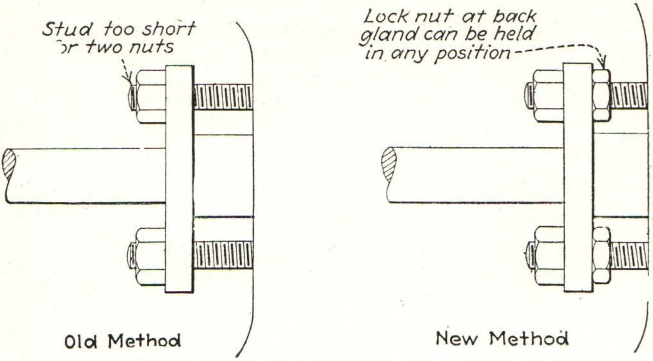

Stud too short for two nuts
Lock nut at back gland can be held in any position
Old Method
New Method

Fig. 2—Lock nuts used on packing glands of air compressor

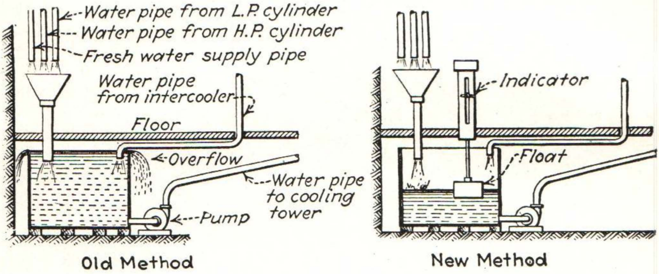

Water pipe from L.P. cylinder
Water pipe from H.P. cylinder
Fresh water supply pipe
Water pipe from intercooler
Floor
Overflow
Water pipe to cooling tower
Pump
Indicator
Float
Old Method
New Method

Fig. 3—Tank for receiving cooler water

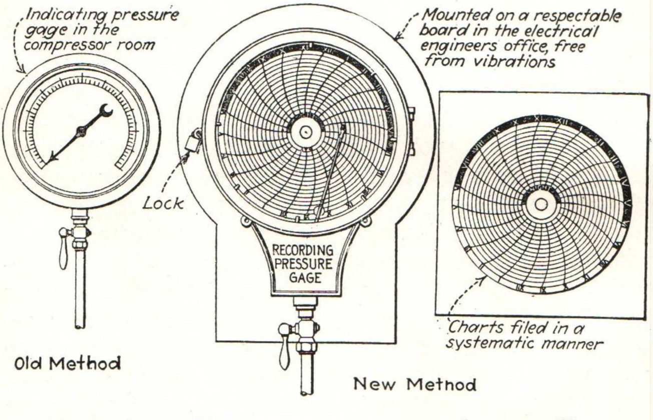

Indicating pressure gage in the compressor room
Mounted on a respectable board in the electrical engineers office, free from vibrations
Lock
RECORDING PRESSURE GAGE
Charts filed in a systematic manner
Old Method
New Method

Fig. 4—Recording pressure gage suplants ordinary pressure gage

pans and splash guards constructed to fit all necessary places about the machine. Oil caught was filtered and used again in places where high-grade oil was not essential.

The next feature was to reduce the length of the pipe discharging water from the intercooler. The purpose of this was to enable the compressor attendant to see the stream of water discharged so that any interference with the water flow could be readily detected. Fig. 1 represents the condition as remedied.

The piston-rod gland on the high-pressure crank end was held in place by single nuts as the stud bolts were too short to accommodate lock nuts. As a consequence the nuts worked loose when the gland was loose enough to prevent the packing from burning. When the gland was tightened sufficiently to prevent leakage, the packing would be burnt. The difficulty was overcome by placing thin nuts back of the packing gland as shown in Fig. 2. In this way the packing gland could be held in place with sufficient pressure to overcome leaking but loose enough to prevent undue pressure upon the packing.

The water discharge from the intercooler and the cylinders was collected in a tank under the compressor-room floor and returned to the cooling tower by a small centrifugal. When the amount of flow exceeded the capacity of the centrifugal pump, the water in the tank would rise and overflow. The tank was fitted with a float indicator as shown in Fig. 3. This enabled the attendant to know at all times the height of water in the tank and to regulate the amount of flow accordingly.

A recording pressure gage was installed, and the first few records indicated the dubious condition of the compressor. The air pressure, because of faulty operation of the unloader, fluctuated through a range of 30 lb. Operating pressure was 90 lb. Wide range in air pressure caused trouble and lost time at the drills. The recording gage gave a complete record of compressor operation and eliminated controversies between mine and surface plant operators. A new unloader, Fig. 5, was devised to remedy the pressure variation. By experimentation, the counterweights, adjusting levers, dashpots and other details of the unloader were brought

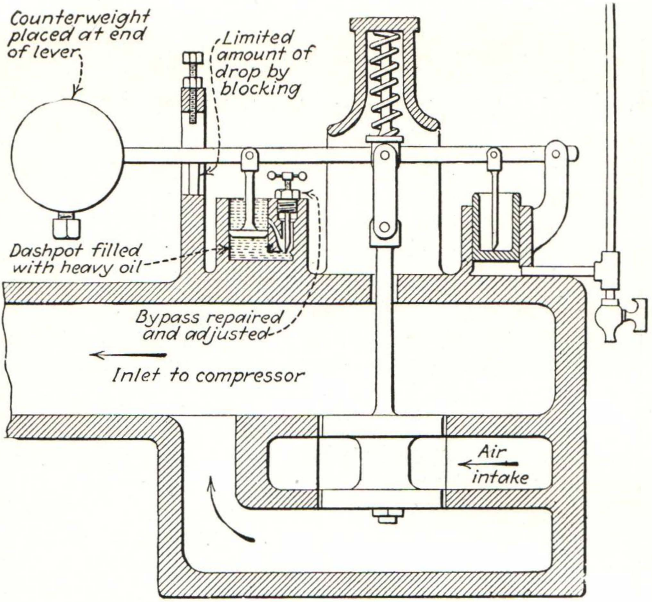

Counterweight placed at end of lever
Limited amount of drop by blocking
Dashpot filled with heavy oil
Bypass repaired and adjusted
Inlet to compressor
Air intake

Fig. 5—Unloader rig on air compressor

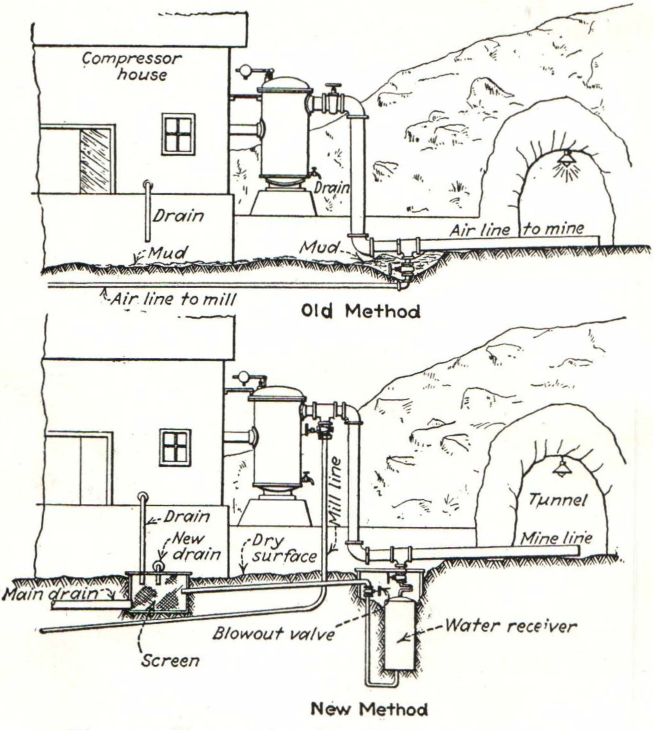


Fig. 6—Change in air piping arrangement

into line. It was found that the best results were obtained by the use of a heavy oil in the dashpot and an accurate bypass adjustment. The counterweight was placed at the end of the lever and the amount of drop of the counterbalance was limited as shown in the figure. The result of this work was to reduce the pressure variation to reasonable limits, five pounds above or below 100 lb. pressure.

Another operating difficulty was the excess moisture in the compressed air. An investigation to determine the presence of leaks by which cooling water could get into the system indicated the real cause of trouble, which was in the position of the air line to the mill. This was cut into the air line to the mine at the lowest point and received the condensed water which accumulated in the pipe line. The condition is shown in the upper sketch of Fig. 6. The piping was redesigned as shown in the lower part of Fig. 6. A water receiver consisting of an old hot-water boiler was placed at the low point of the mine air line and served to remove all of the water. The mill air line was cut into the main close to the receiver.

Trucks and Small Electric Shovels at United Verde

In the large open pit of the United Verde Copper Co., Jerome, Ariz., four raises have already been put up from the lower levels and several more will soon break through. The glory-hole method is used wherever practicable, but the bulk of the ore will be handled with trucks and electric shovels. With this elastic system of handling the ore, a number of products will be mined and kept separate. For example, the waste will be loaded into trucks and taken to the waste raise, the high sulphide ores to the sulphide raise, the siliceous ores to the silica raise, and the milling ore to the concentrator raise. The small electric shovels are used to load the trucks and the trucks which do not require any track-laying tram the ore to the various chutes without delay. This arrangement furnishes a very elastic system for selective surface mining.

Mine Dewatering Diagram

By W. M. Hutton

3 London Wall Buildings, London, E. C. 2

Dewatering a flooded mine may be represented graphically on a time base to show the effect of varying rates of inflow and of varying rates of pumping, and the diagrams are particularly useful where these rates increase or decrease as the water is lowered. By "rate" is meant quantity per hour or other unit of time. The important factors are (a) total quantity in mine or "dead" water; (b) rate of inflow of new water as the water level is lowered; (c) capacity of the dewatering plant at various stages of the operation.

The accuracy of estimate of factor (a) will vary widely in different cases. Sometimes it will be little better than a guess based on what ore is believed to have been shipped from the mine plus the dumps found on the surface. On the other hand, where accurate plans of underground workings exist, the quantity of "dead" water can be closely estimated. Correct predetermination of factor (b) is of great importance. Though an underestimate of "dead" water would only entail an extension of the time necessary to dewater the mine, an underestimate of the inflow might protract the operation indefinitely and call for increased pumping plant and starting all over again.

Furthermore, a record of what the mine was finally "making" at a certain depth may not be a safe basis of estimate as to what is to be expected in dewatering it at a more rapid rate than that at which it was opened up originally. This point is important, although not always realized. Inflow is not necessarily the result of an underground spring of constant flow. It may be the result, partly at least, of pervious country draining into the workings as the mine opens up. This opening up may have extended over years, and the quantity to be handled divided over this period would represent a rate of flow much less than from water-logged country being rapidly drained a second time. Put in another way, if approximately the same total amount of inflowing water is to be handled in a shorter time, the rate of pumping must be proportionately increased.

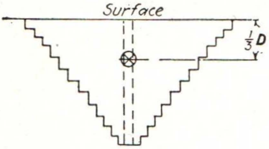


Fig. 1

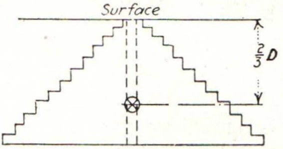


Fig. 2

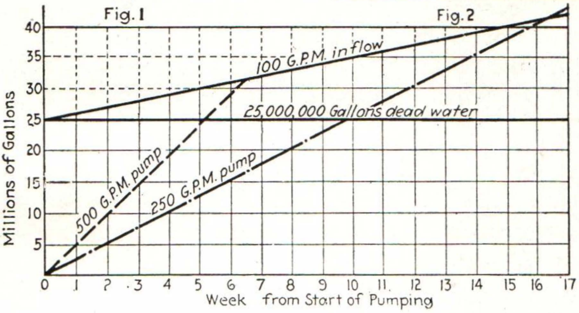


Fig. 1—Showing higher center of gravity when mine openings decrease with depth

Fig. 2—Showing lower center of gravity when mine workings widen in depth

Fig. 3 (Lower)—Diagram showing "dead" water augmented with constant influx

This consideration should be kept in mind in using previous records of inflow.

The time factor (c) will be all-important in deciding the capacity of the dewatering plant. The shorter the time allowed the greater will be the plant capital cost but the less will be the operating cost as represented by wages and fuel. This is clearly indicated in Fig. 3, where the 500-gal. per minute pump is shown not only to have emptied the mine in half the time but also to have handled a considerably less *total* volume of water than the 250-gal. unit. The total power will, of course, be proportional to the total water raised. Ample capacity, then, is not only prudence, in view of the uncertainty of factors (a) and (b), but is in the direction of economy in total cost. The plant may be of a type which, granted continuous operation, will maintain a constant rate of delivery throughout the operation, or it may be of a type which, no matter how efficiently operated, will steadily fall in capacity as the water level falls. Constant-speed reciprocating pumps would be in the first category; air lifts and hoisting by bailing skips would be in the second. It is obvious that at a fixed rope speed a bailing skip will raise nearly double the quantity per hour from 500-ft. depth as from 1,000 ft. The relative merits of these systems should be considered in conjunction with a diagram showing the distribution of the water in relation to depth. Figs. 1 and 2 illustrate different conditions. In Fig. 1 the average depth from which water is to be raised is only one-third of total depth. In Fig. 2 the average lift is two-thirds of total depth. The ratios correspond to the center of gravity of the figures represented.

For a case resembling Fig. 2, a plant which maintains its full capacity right to the bottom of the mine would be indicated, while bailing or air lifting would show to advantage in Fig. 1. These diagrams represent "dead" water only. Where a compound diagram is constructed to include inflow, the tendency will be towards the condition represented in Fig. 2. Fig. 3 shows the elementary case of a fixed quantity of "dead" water with an inflow of constant amount from the start superimposed upon it. It incorporates these various factors in such a way that the time taken in the operation is arrived at graphically, and alternative methods and rates of pumping can be directly compared. The mine is dewatered on the date where the pumping curve or line cuts the quantity line. Up to this point the conditions are so simple that diagrams should be unnecessary. If, however, as is more probable, the rate of inflow increases as lower depths are reached, the total quantity line will no longer be straight, but will be curved to a degree varying with the rate of pumping. In other words, if pumping capacity is such that the bottom is reached in a month, the maximum rate of inflow will also be reached on that date. On the other hand, if the operation takes six months the maximum inflow will also be delayed to that date. To be correct, then, a series of curves should be plotted, each representing rate of inflow for different times to empty the mine. Such curves plotted for one, two, three, and four months' operation are indicated on Fig. 4. As previously noted, the pumping line or curve on Fig. 3 is for an undiminished rate of delivery throughout, and, consequently, straight. Delivery by bailing

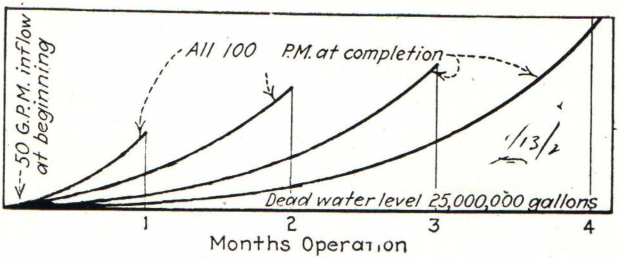


Fig. 4—Showing increased amount of water to be handled when inflow increases from 50 to 100 gal. per minute

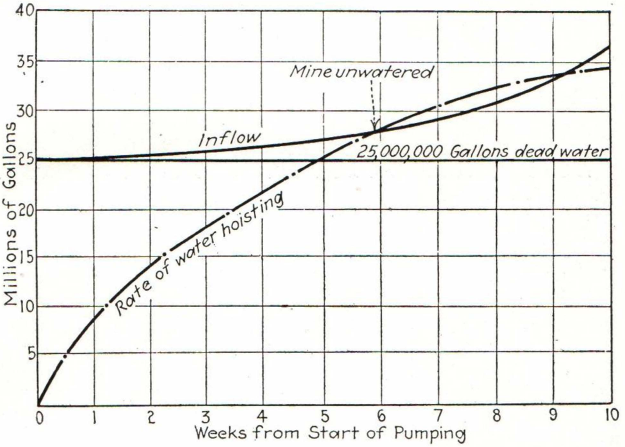
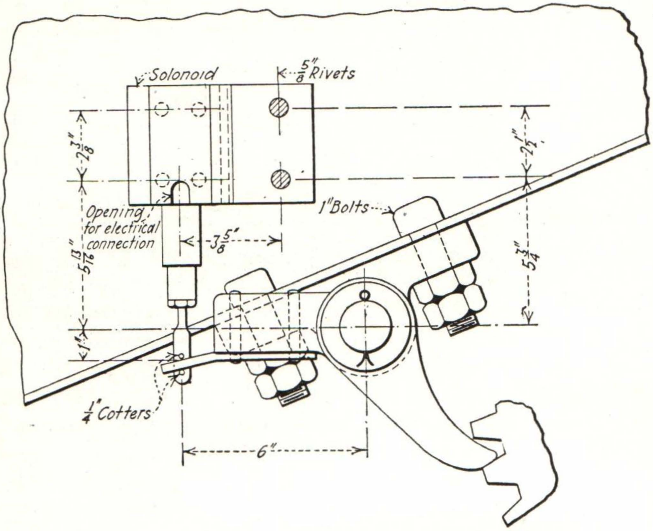


Fig. 5—Showing prolongation of pumping time when inflow increases and rate of dewatering decreases with depth

skips will, however, be represented by a curved line as shown by Fig. 5, indicating how capacity falls off as the water level falls. The diagram corresponding to Figs. 1 and 2 would be used in plotting this curve. One great value of the diagram is that it indicates in a striking manner the joint effect of increasing inflow and decreasing pumping rate as the lower depths are reached. It is not too much to say that here lies the cause of many a failure and many a weary extension of the time necessary to unwater a mine.

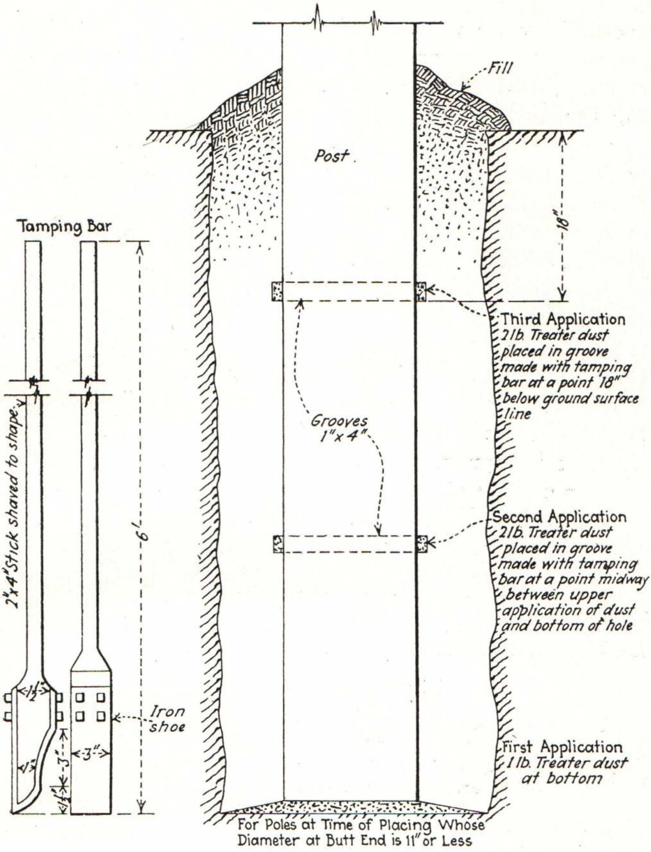


Solenoid Lock for Incline Conveyors

Some kind of a locking device is necessary at the head of an incline conveyor belt run to prevent the loaded belt from reversing its direction of travel when it is stopped. At the Utah Copper Co.'s plants a conveyor lock operated by a solenoid was first used for this purpose on belt-driven inclined conveyor units. Subsequently when gear-driven units surplanted the belt drives a solenoid brake was applied to the motor shaft and is now standard. However, the first device is a satisfactory one for belt drives. It consists of a pawl which is pivoted upon a short shaft carried by a bracket at the conveyor head. The pawl engages in the driving gear. When the motor is stopped the solenoid automatically comes into action and throws the pawl into the main gear. When this stops the pawl locks into the gear, preventing any reversed motion. The solenoid coil is carried upon a bracket. Details of the arrangement are shown in the accompanying illustration.

New Timber Preservative

The use of cold treater dust, having a high arsenic content, as a timber preservative where timber such


Fill
Post.
18"
Tamping Bar
2"x4" Stick shaved to shape
6'
1½"
1"
3"
3"
1½"
Iron shoe
Grooves 1"x4"
Third Application 2 lb. Treater dust placed in groove made with tamping bar at a point 18" below ground surface line
Second Application 2 lb. Treater dust placed in groove made with tamping bar at a point midway between upper application of dust and bottom of hole
First Application 1 lb. Treater dust at bottom
For Poles at Time of Placing Whose Diameter at Butt End is 11" or Less

Preserving the wood in a new pole. The tamping bar used

as transmission poles, fence posts, and other wooden structures coming in contact with soil are to be preserved has recently been developed by the Anaconda Copper Mining Co. The method has the advantage of low cost. Apparently the dust sterilizes the soil in the immediate vicinity of the wood and penetrates the wood as well. Its slow solubility prolongs the beneficial action. It is placed at the bottom of the pole and in one or more rings immediately surrounding the pole. As the dust is poisonous, it is necessary to prevent it from getting into cuts, open sores, or scratches. Gloves should be worn when handling it and care taken to avoid its getting into eyes, nose, or mouth. Water which may be in the pole hole should be bailed out. In applying the dust to a pole in place, the ground is excavated to a depth of 25 in. and the pole freed from all decayed wood. The hole is then backfilled to a depth of 20 in. from the ground line. A special tamping bar is then used to make a groove 1 in. wide and 4 in. deep completely around the pole at this depth. The groove is filled with 2½ lb. of treater dust. Fresh dirt is then placed for an additional depth of 10 in. without tamping and then backfilled to a point 10 in. below the ground line, when another ring of dust is placed as in the first instance. Fresh dirt for a distance of six inches is placed above the second ring of dust, and the hole backfilled and tamped. In placing new poles one pound of treater dust is placed in the bottom of the hole and two rings, each of two pounds of treater dust, are placed above the bottom as shown in the accompanying sketch.

Scraper Loaders, as Designed and Used at Casapalca, Peru

BY C. S. T. FARISH AND J. C. PRATT

Assistant Superintendent and Chief Engineer, respectively,
Cerro de Pasco Copper Corporation

THE DESIGN OF THE FIRST LOADER made at Casapalca, Peru, was taken from that shown in the *Engineering and Mining Journal* of May 3, 1924, as developed by the Mineral Mining Co., of Iron Mountain, Mich., and called the Osana loader. Of this the most important feature was the use of a cantilever boom for the dump, which gave a self-contained unit, as compared with the earlier slides built in several sections and requiring considerable time to set up and dismantle. The Cerro de Pasco Copper Corporation's mining department prepared the plans for this, the first Osana loader in Peru, and, upon its recommendation, the Sociedad Minera Backus y Johnston del Perú obtained and used them.

The loader was put in operation on the 2,100 level of the Carlos Francisco mine in a 7x9-ft. crosscut of some length. As the crosscut was straight, the large size of the loader did not cause much difficulty, but when the vein was encountered, with its twists and turns, even though the same section of heading was used, the loader became very awkward to handle.

Moreover, the box-type scraper with saw teeth and a square yoke across the front was not suitable for the rock, which was often coarse. On the whole, the loader was not a success in work on the vein.

IMPROVEMENTS ON LOADER

However, the possibilities were recognized, and R. M. Sánchez, the former engineer, designed a second loader for use in a new tunnel, the cross-section of which was to be about 8x9 ft., with a large ditch on the side. In principle, the loader was the same as the first one, with the following changes: The frame was made square in section, which raised the position of the hoist considerably, but still kept it below the height of the rear sheave. The over-all dimensions were reduced on all sides. Instead of a sharp break in the slide, an arc of 9-ft. radius was used. (It was found that even with this arc the bottom plate of the slide was worn quickly by the rope, so a half-inch reinforcing plate was put over this part, and changed as needed.) Six-inch channels were used for the sides of the slide, with the flanges inside so that the channels could be riveted to the frame. It may be noted here that the flanges caused the scraper to catch sometimes, which led to the use of angles in their place on the later designs.

This second loader was designed for use with a 1½-ton car of the dimensions given in Table I. A box-type scraper was first used, but this was later replaced by one of the hoe type, which has since been changed repeatedly in design until a satisfactory one has been developed. This loader gave excellent results in the tun-

Table I—Comparison of Sizes of Loaders

	Over-all Length,		Width of Frame,		Width of Slide,		Maximum Height,		Width of Lip,		Size of Car Used,
	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	Tons
Original Osana...	17	0	4	0	3	7	6	0	5	6	1½
Second loader....	14	10	3	8	3	3	5	8	4	6	....
Type D loader...	12	5	3	0	2	8	5	11	4	0	1½ and 2½
Type A loader...	10	10½	3	0	2	8	5	6	4	0	1

nel until it wore out, but it was considered too large for use in small headings.

Loader for Universal Use Sought

Since by this time the feasibility of the Osana loader had been thoroughly proved, all that remained was to develop one suitable for use in all places. Immediate requirements called for a small loader for use with a 1-ton car in old workings that were narrow and tortuous. The Type A loader was designed to meet this need, which it has done satisfactorily. At present, however, our most important workings are those using gable-bottom 2½-ton cars, for which the larger Type D loader was designed.

The veins in the mines at Casapalca are narrow and crooked. Headings are usually 6x8 ft. in the clear on all but main haulage levels using 2½-ton cars, on which they are 7x9 ft. However, regardless of the size, the many turns in the vein make a short loader essential, and, at the same time, one that has a minimum width commensurate with ample capacity of scraper.

Reference to Table I will show that the size of loaders has been reduced to the least possible dimensions for the cars used with them. Small size not only facilitates moving the loader from one place to another, or to and from the face, but it also allows free passage at the heading. Either Type A or Type D, when dismantled, can be put on a mine cage. Dismantling consists in taking off the front lip (two bolts), removing the boom (ten bolts on each side), taking off the trucks (two bolts for each wheel), and removing the hoist.

Table II—Dimensions of Cars

(Size of Box)

	Length,		Height from Track		Width,	
	Ft.	In.	Ft.	In.	Ft.	In.
1-ton	3	8	3	8	2	3
1½-ton	4	0	4	0	2	8
2½-ton	5	0	4	0	3	0

Details of Construction

In the first loaders the boom was bent on an arc at the frame, but since the bending of the angles involved a great deal of work, it was decided to continue them straight, cutting off the short leg at a point such that the bottom plate of the slide could be bent downward.

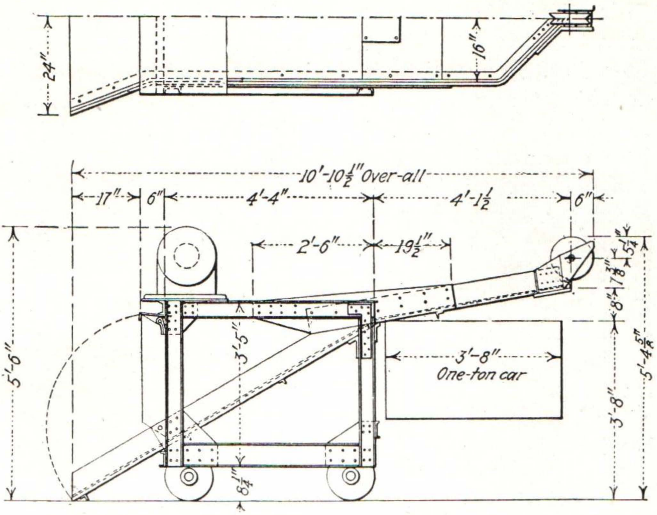
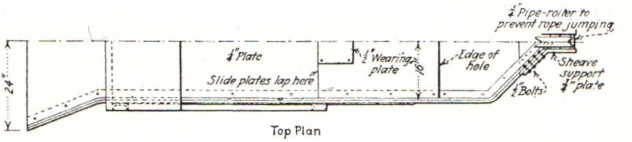
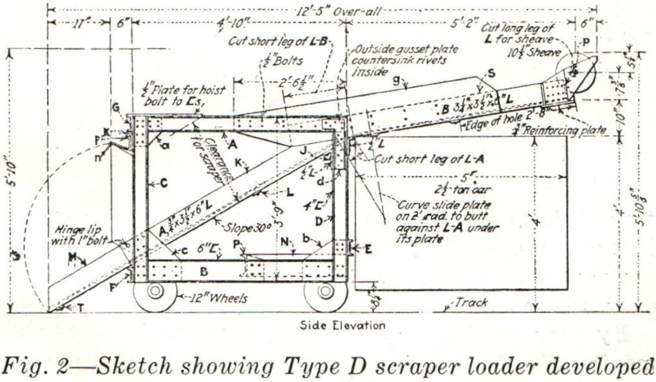


Fig. 1—Sketch of Type A scraper loader developed at Casapalca, Peru



Top Plan



Side Elevation

Fig. 2—Sketch showing Type D scraper loader developed at Casapalca, Peru. The letters refer to the details in Fig. 4

This gives, also, a strong reinforcement for the large gusset plate. The short leg of both members *A* and *B* (see Fig. 2) should be cut about 4½ in. from the point at which the lower edges of these two angles meet. By so doing, the slide plate of the boom can be curved in an arc and fitted under the plate of member *A*, which should project ¼ in. beyond the point where the short leg has been cut. Thus the ½-in. wearing plate, already mentioned, should not come to the edge of the plate, but this distance back of it. It should be beveled to a knife edge at each end so that the scraper will not catch under it.

The entire slide is made of ¾x3½x6-in. angles with the long leg vertical. This heavy angle has been adopted after having had a lighter one buckle completely at the bend in the rear.

Inside width of the slide has been reduced to 2 ft. 8 in., which has been found sufficient for all purposes. Standard width of scraper has been set at 2 ft. 6 in. over all, thus allowing 2 in. clearance on the slide. This is ample; in fact, greater clearance will result in jamming the scraper by rocks that fall from it.

The yoke of the scraper is bent at 45 deg., which prevents it from catching on any obstructions in its path; of these there are usually many. At the same time, this allows the rear end of the loader to be bent in the same way, so that its over-all length is that at the sheave only, and its effective length in going around curves is much less.

Table III—Time Studies of Hand and Loader Operations

(Car holding 2⅓ tons)

	Mucking		
	Minutes per Car	Man-Minutes per Car	Man-Minutes per Ton
By hand	15.30	67.60	28.95
By loader	6.83	32.95	14.40

Time per Car for All Operations—Minutes

	Hand	Loader
Mucking	15.30	6.83
Tramming	12.10	10.40
Loafing	1.70	1.10
Total	29.10	18.33

By hooking the hinged front lip to the frame when moving the loader, the length is further reduced by some 17 in., and as the track level is usually the place where are most of the narrow places and obstructions, such as pipe lines, the hinged lip is a decided advantage.

The axle of the sheave should be so secured that it cannot turn. The sheave should have a deep groove, with plenty of flare to the flanges so that the rope will not jump out with the constant jerking as the scraper goes through the muck pile. By putting a pin across the ends of the two supports with a pipe around it, a great deal of this trouble may be eliminated. Because the sheave for the head block is inclosed in plates, there is no difficulty with it from this source.

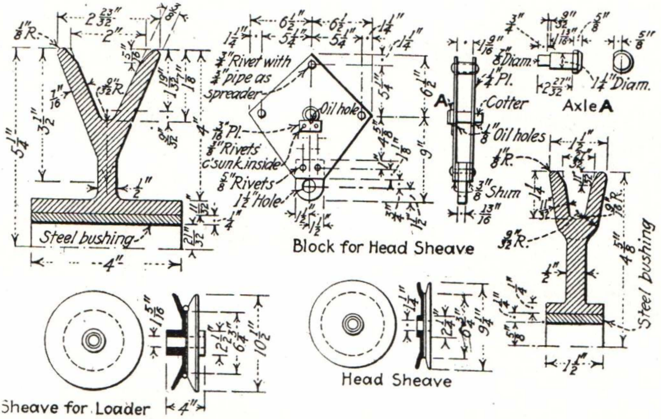


Fig. 3—Detail of sheave for Osana loader

Owing to the heavy wear on the sheave and its supports, bolts should be used in fastening the supports to the boom, so that they may be readily taken off and repaired when necessary. The long legs of the angles of the boom must be cut away sufficiently at the point of the V to allow the sheave to enter.

The size of the base plate for the double-drum hoist is determined by the type of hoist used. In all cases, however, clearance must be allowed for the scraper to pass below it. For the scraper used with it this be at least 22 in. and preferable 24. It is this clearance that makes it necessary to project the plate beyond the frame on the Type A loader. The plate is secured to the flanges of both the side and front channels by as many bolts as are deemed necessary—about twelve.

An outside gusset plate is attached to the upper rear

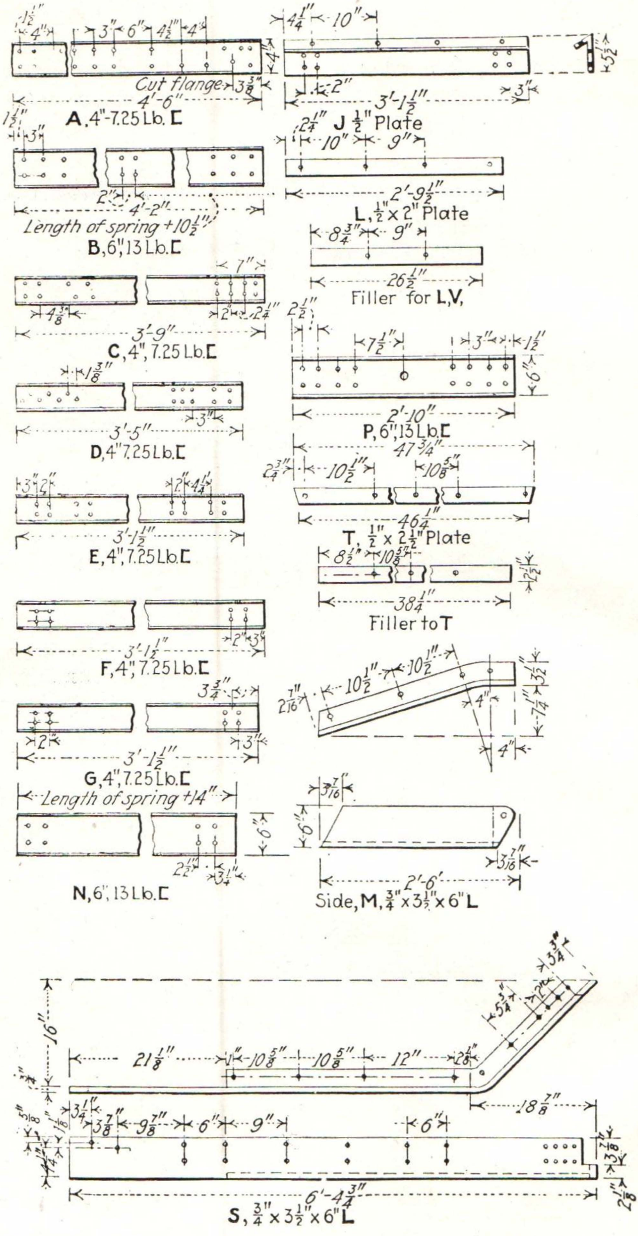
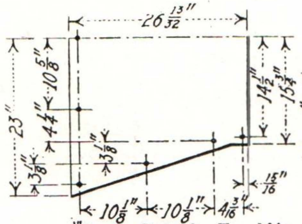
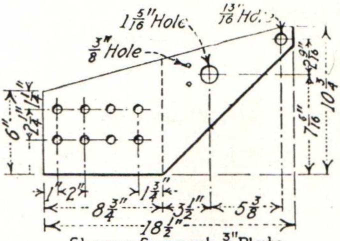


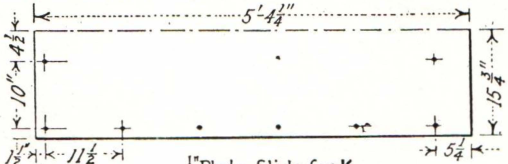
Fig. 4 (a)—Details of Type D scraper loader. See Fig. 2



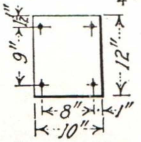
1/4" Plate, Side for Front Lip



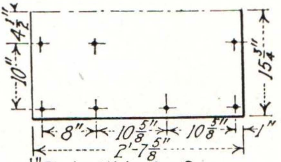
Sheave Support, 3/4" Plate



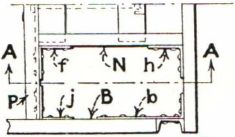
1/4" Plate, Slide for **K**

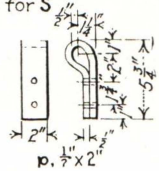


1" Wearing Plate for Slide

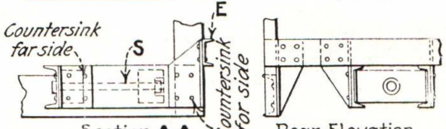


1/4" Plate Slide for **S**



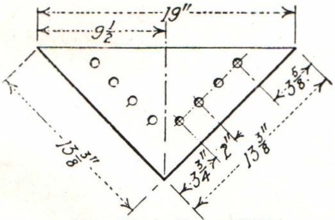


p, 1/2"x 2"

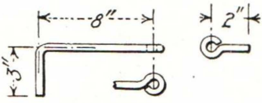


Section **A-A**   Rear Elevation

Assembly of Car Bumper

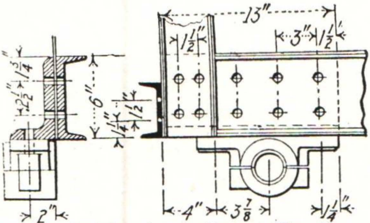


m, Base Plate Rear of Slide 3/4" Plate

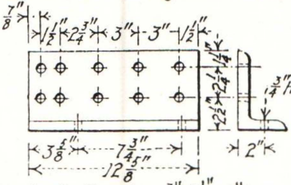


n, Hook for Lip 1/2" Round

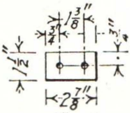
Fig. 4 (b)—Details of Type D scraper loader. See Fig. 2



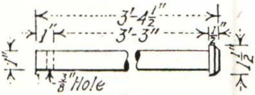
Attachment of Bearing to Frame (Front)



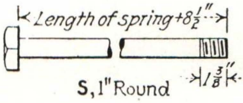
Angle for Bearing ¾"x3½"x6"**L**



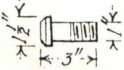
r, ½" Plate



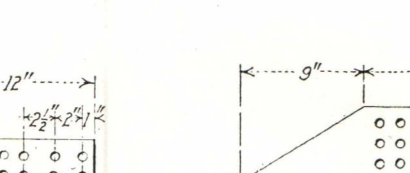
K, 1" Round

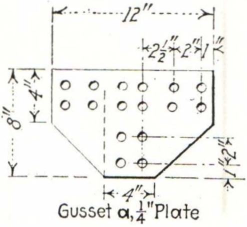


S, 1" Round

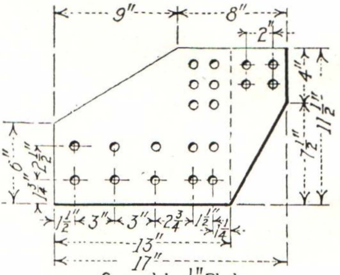


1, 1" Round



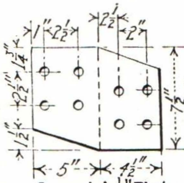


Gusset **a**, ¼" Plate

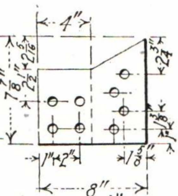


Gusset **b**, ¼" Plate

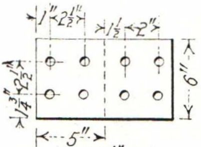
Fig. 4 (c)—Details of Type D scraper loader. See Fig. 2



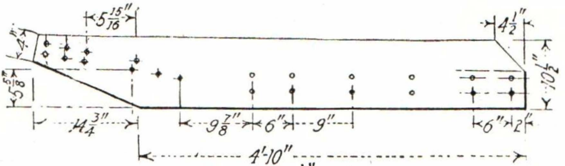
Gusset **j**, ¼" Plate



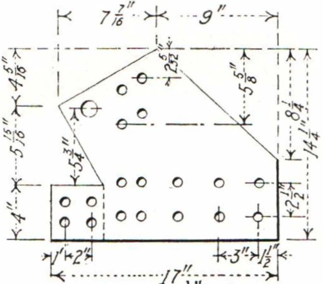
Gusset **e**, ¼" Plate



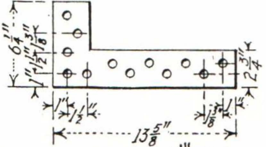
Gusset **f** ¼" Plate



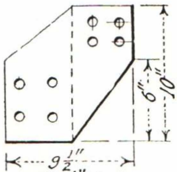
Gusset **g**, ¼" Plate



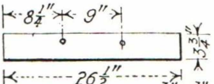
Gusset **c**, ¼" Plate



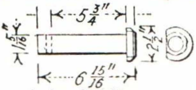
Gusset **d** ¼" Plate



Gusset **h**, ¼" Plate



Filler at Bottom of **K** ¾"x3¾" Pl.



Axle for Sheave

Fig. 4 (d)—Details of Type D Scraper Loader. See Fig. 2.

corner of the frame, with rivets countersunk inside, to form the joint between the horizontal and vertical channels. The axles of the wheels are carried on outboard bearings attached to the lower channels of the frame. A bumper may be put in the frame to prevent the cars from deforming the rear channel when they are pushed under the loader. A roller is put under the front top channel of the frame as an idler for the rope when the scraper is on a high muck pile.

TYPE OF SCRAPERS

The large number of scrapers that have been developed in recent years all over the world makes it almost useless to go into detail on this subject, but as we had to do a great deal of experimenting before we found one that gave satisfactory results for all classes of work, it will be well to outline the salient points of the model finally developed.

The small teeth on the scraper are reversible, so that they may be turned end for end. Under them is a removable plate that protects the main plate of the frame from wear. The long teeth are of a single piece, but can be taken off when worn. These also could be made reversible by drilling two rivet holes on the other leg and changing the shape so that it is symmetrical: but they are so little used that this expedient is hardly necessary.

The feature of the curved yoke, already mentioned, cannot be too strongly emphasized. It is especially noticeable when the scraper comes in at an angle to the loader, which it will do in a great many cases. This scraper is simple in design and easy to construct, but can still be greatly improved as ideas suggest themselves with experience.

USE OF LOADERS

Outside of the use in headings, the loaders have proved of great value in cleaning up cutting-out stopes, in which the ore is broken down on the track in preparation for stulling and shrinking. The resulting muck pile is about 15 ft. high and as long as the stope. As soon as the blasting has been advanced sufficiently so as not to injure the loader, mucking is started. The loader is placed at the edge of the pile and advanced once a shift. As the Type A loader will load a 1-ton car in from three to four minutes, with native labor, the speed depends mostly upon the number of carmen that are employed and the distance of the loader from the chute.

One of the loaders is working in a large tunnel in which the track is carried on stulls 3½ ft. above the bottom. Here a slide is put up to the level of the track and ballasted before blasting. The loader handles over 100 tons of rock in about five hours, loading into 1½-ton cars, or, in approximate figures, 20 tons per hour.

CONCLUSION

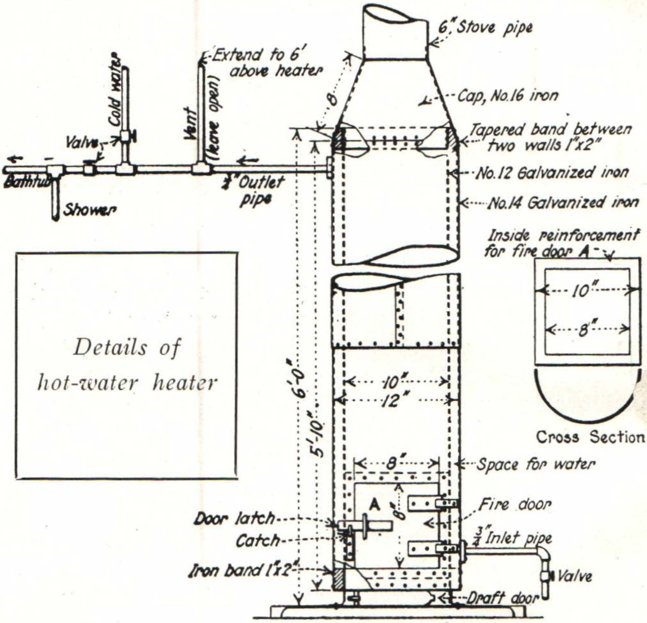
To date, we have in use six loaders at Casapalca, and are building others. Six have been made for the Morococha Department of the Cerro de Pasco Copper Corporation, and the use of twelve more is under consideration. One has also been asked for a power-tunnel project of the same company. In addition, two larger loaders of the same general design were built for a railroad tunnel and proved successful, though no doubt a mechanical loader would have been better had the size of the job on which it was employed warrant it, which it did not.

The cost of building one of these loaders at Casapalca is about $350, complete with scraper. To this must be added the cost of a double drum hoist. This figure should be much lower in the United States in view of the different conditions obtaining there.

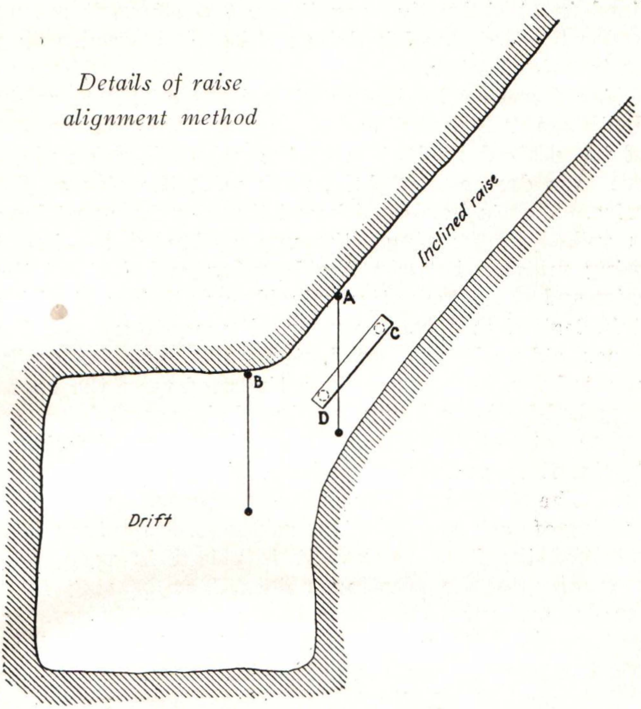
Though it may be that others have developed the Osana loader along similar lines, the types herein described have been worked out entirely in Casapalca. We feel greatly indebted to the originators in having given us the basic principles upon which to work, for we have done nothing more than change and modify until we had a loader to suit our conditions, and one which, we believe, will be found to have a wide range of use elsewhere.

Hot-Water Heater

A simple type of hot-water heater that can be made in the mine shop will serve to provide hot water for a tub and shower or two showers, as shown in the accompanying figure. The heater was designed and is in use at the Plymouth mine, Plymouth, Calif. It supplies hot water for the mine foreman's office. The heater is constructed of an inner wall, No. 12 galvanized iron, and an outer wall, No. 14 galvanized iron. The two walls are separated by a 1-in. water space. At the top the inner and outer sheets are riveted to tapered iron ring, 1 in. in thickness. A bottom ring, 1x2 in. in section, is placed between the sheets at the bottom and a frame forms the fire door. A thin sheet-iron frame, with draft door, is placed below and furnishes the support of the heating tank. A conical cap of No. 16 iron is riveted to the top and connects with a 6-in. stove-pipe. All of the joints are riveted and soldered. A $\frac{3}{4}$-in. water pipe is tapped into the bottom and at the top of the water space a pipe of the same size connects with the plumbing. A vent pipe extending to a height of 6 ft. above the top of the water space provides for the escape of steam and guards against excessive pressure. A small wood fire suffices to heat the water quickly.



Details of hot-water heater



Details of raise alignment method

Aligning Raises for Connections

By G. H. Parker

Brule Mining Co., Stambaugh, Michigan

The method of aligning raises to be described applies equally well to raises with inclinations ranging from 0 (horizontal drift) to 90 deg. It requires a small amount of engineering work. When the lines are once in place they are usually permanent and may be used by the workmen as well as the engineer.

A transit and a protractor level are required to fix the controlling lines. The Brunton compass serves the latter purpose, and, in short raises, may be used with reasonable accuracy in place of the former for fixing bearing lines (provided the mine is free from local magnetic disturbances). For fixing bearing lines, a spad is placed at *B* (see accompanying figure), either in a wooden plug or in drift timber. Another spad is then lined in at *A* by the transit, with the transit set up under *B*. This spad is driven either in a wooden plug or in raise timber.

For directing inclination, two holes a foot or so in depth are drilled at *C* and *D* approximately on the line of the desired inclination of the raise. Wooden plugs are then driven in tightly, flush with the side of the raise. Then a hardwood board three or four feet long is spiked to the plugs, the upper edge of the board being midway between the hanging and foot of the raise and at an inclination coinciding with that at which it is desired to drive the raise. By sighting along the upper side of this board, the raise may be easily directed for inclination. In a timbered raise the board is spiked to the cribbing, no plugs being needed.

One advantage of this method is its simplicity. Again, it is permanent and always ready for instant use. The miner can check his work before starting the drilling of each cut. Blasting seldom destroys the alignment of the board unless it is placed in very close proximity to the blasting point. The bearing lines (cords) are hung aside on a hook while not in use. In a timbered raise the "inclination" board may be kept within a short distance of the working face.

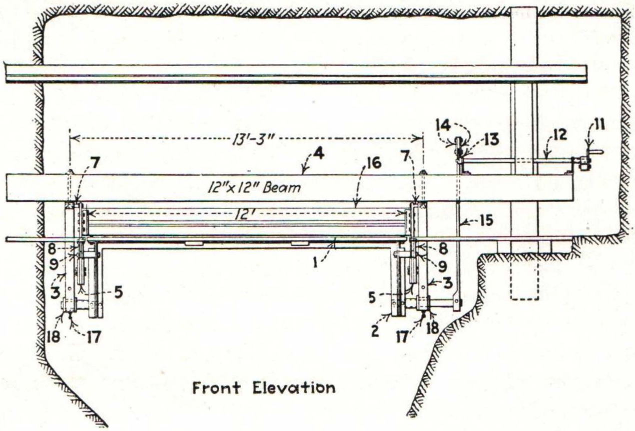
This method of aligning raises has been used with decided success for "holing through" raises up to 160 ft. in length, and its accuracy, if checked occasionally, should insure success in much longer work.

Waste and Ore Transfer Gate

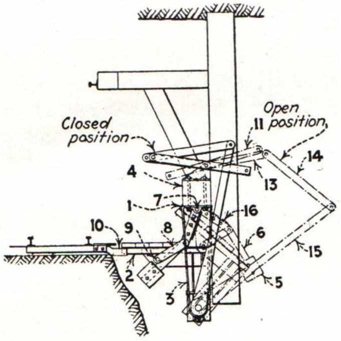
By C. S. Watt
Assistant Mechanical Engineer, Dome Mines, Ltd.,
South Porcupine, Ontario

Receiving openings of waste and ore transfers or bins, require protection, and this is afforded best by platform doors, grizzlies, or other devices. At the Dome mines, at South Porcupine, Ontario, a platform door was designed and installed for this purpose. It has been in use for about a year and has proved safe and easily operated. The illustration shows the front and side elevations with the platform door closed and also in the open position.

The door platform is 12 ft. long, 30 in. wide, and



Front Elevation



End Elevation

Ore transfer gate at Domes mines

3 in. thick, and is constructed of fir. It is secured with bolts to an angle-iron frame, 2, the frame being pivoted on two hangers, 3, which are fastened to a 12 by 12-in. beam, 4. The door is balanced by two weights, 5, one at either end attached to the bent bars, 6. The bent bar is pivoted on a pin, 7, which is fastened by the same bolts that support the hanger. Between the balance-weight bar and the door frame, 2, are two links, 8. In the closed position the weights assist in opening the door, and as they swing past the center line they also assist in closing the door.

In the closed position the balance-weight bar bears against the pin, 9, to which the connecting links are attached to the door frame, thus acting as an additional stop for the door besides the two stops, 10, that are fastened under the track rails.

The door is operated by a counterbalanced lever, 11, fastened to the operating shaft, 12. Another lever, 13, is fastened to this shaft, two connecting links, 14, being attached to the long door frame lever, 15. To open the door the counterbalanced hand lever is turned 180 deg. to the right, when the door stops against the 12 by 12-in. beam.

A curtain plate, 16, is fastened with bolts to the two balance-weight bars so as to close the space between the 12 by 12-in. beam and the platform door. This is necessary where the transfer or chute is near a crusher station. By this arrangement the chute is closed in and the station kept free from dust. To level the door, jack screws, 17, are tapped into the hangers, 3, and the bearings, 18, are adjusted to suit the conditions.

Doors are made in 12 and 14-ft. lengths and are so balanced that very little effort is required to close and open them, either operation taking about two seconds. The hand-operating lever can be replaced by a chain wheel and operated by an endless chain, supported on idler wheels, above the track where it is necessary for the motorman to open the door as he approaches the transfer.

Maintaining Shaft-Plumbing Bob in Position

By Edward K. Judd
Mining Engineer, Palisade, New Jersey

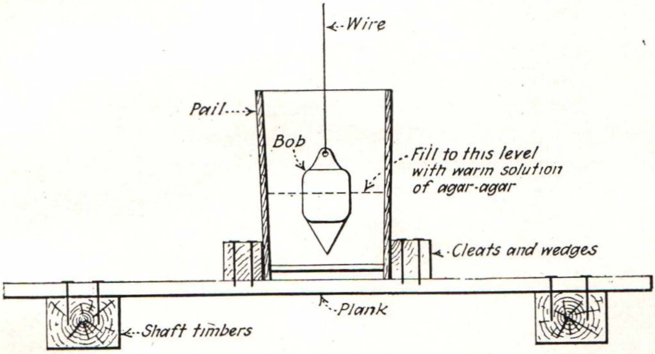
As a means of avoiding the tiresome and painstaking process of determining the mean position of a swinging shaft-plumbing wire, and the usually inconvenient or unsatisfactory methods for fixing it in that position, when found,[1] the following suggestion is offered:

After suspending the wire, with the customary precautions to insure that it is free and as little affected as possible by air currents and falling water, set an empty can or a small pail on a firmly fastened plank and adjust its position, as shown in the accompanying sketch, until the plumb bob just clears the bottom and all sides; a large amount of clearance is not necessary, but it is important that both the pail and the plank are rigidly fixed. Then pour into the receptacle a sufficient amount of a warm solution of agar-agar, previously prepared in another vessel, to submerge about the lower half of the plumb bob. Suitable proportions are about 1½ oz. of agar-agar to 1 gal. of water, and the mixture must be boiled for a few minutes to insure complete solution. On cooling, the liquid passes gradually through stages of increasing viscosity, ending at ordinary temperature as a jelly sufficiently firm to hold the plumb bob quite rigidly in its mean position, at which it will slowly come to rest. If water is falling in the shaft, the solution must of course be protected against splashing and excessive dilution by a split cover having a small hole at its center.

---

[1] Peele's "Mining Engineers' Handbook," p. 1247.

Agar-agar costs about $1.50 per pound at wholesale and about three times as much in drug-store quantities. If much of this work has to be done, the jelly can be dried (it is not sticky or mussy to handle) and used over again. I have not now the facilities for making a practical test as to the accuracy of this proposed method, but it seems that it should compare favorably in many respects with present methods.



Method of fixing plumb bob in shaft surveying operations

Car Locking Device for Shaft Cages

BY A. J. KEAST
Connaught Station, Ontario

Surface and underground shaft station plans at the mine of the Night Hawk Peninsular Mines, Ltd., Connaught Station, Ontario, Canada, are shown in Fig. 1. The station plan provides for the approach of loaded cars on one side of the two-compartment shaft and the removal of the empties on the opposite side. At the surface the loaded cars are removed on one side and the empties are brought in on the opposite side and move on to the cages.

The device for holding the car upon the cage is shown in Figs. 2 A and 2 B. The locking device consists of a ½-in. plate attached to a 1½-in. shaft on either end of which is a throwing device. The plate is of sufficient length to enter between the axles of the car as shown in Fig. 2 B. On the ends of the cage at a height of 2 ft. 9½ in. above the rails are cross bars as shown in Fig. 2 A.

Previous to the installation of this device, which was the idea of our present blacksmith, and of the underground track layout and the surface arrangements, which were the outcome of several employees' ideas after the matter had been discussed, the capacity of this shaft, operating two eight-hour shifts per day and using cars of 1,400-lb. capacity, having at the same time to hoist men and steel and other supplies through the one compartment, was 4,500 tons per month of twenty-six working days. At present, using the same cars, the capacity of the shaft and equipment is 7,800 tons per month. The speed of operation is such that the empty car can be removed and the full car placed on the cage at the loading station in three seconds and the full car removed and the empty placed on the cage on the top deck in four seconds.

When hoisting, the bars on the cage hang upright along the side of the cage. This is effected by means of the long slots cut in the cage bar. The car is held in position by the square arm, C, between the car axles. This arm and the operating bar to which it is attached is so constructed that there is no possibility of its be-

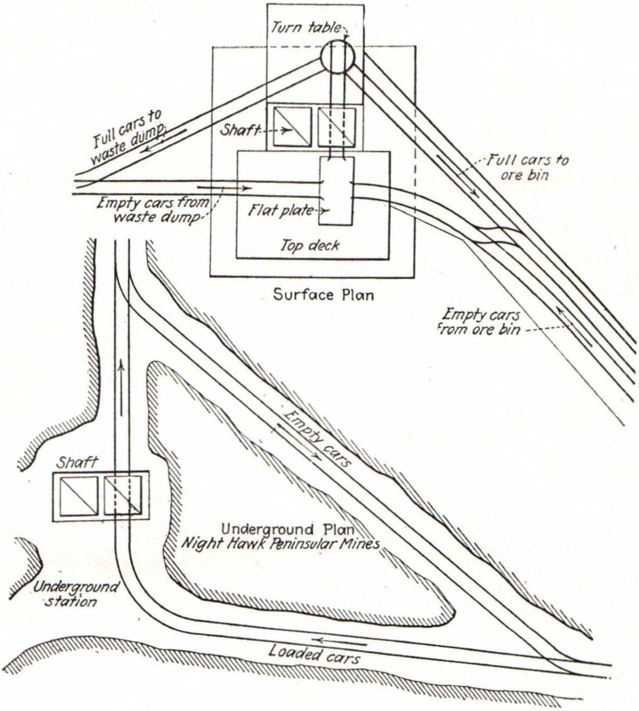


Fig. 1—Station and surface landing arrangements

coming disengaged from between the car axles, its center of gravity being on the same side as the two heavy operating lugs (see Fig. 2B).

The protecting bars on the shaft on the side on which the car leaves the cage are high enough for the car to pass under when they are in place and yet low enough for safety. An additional safety bar, placed halfway between the rail and this high bar, is lowered into place when no hoisting is being done. The cage tender at the loading station has the minimum amount of work to perform, and the topman has time to switch the full cars in the direction of the ore bins or the waste dump, as the case may be, during the time the cage is descending for another full car.

This arrangement, in addition to its safety features, gives a continuous supply of cars. The shaft layout

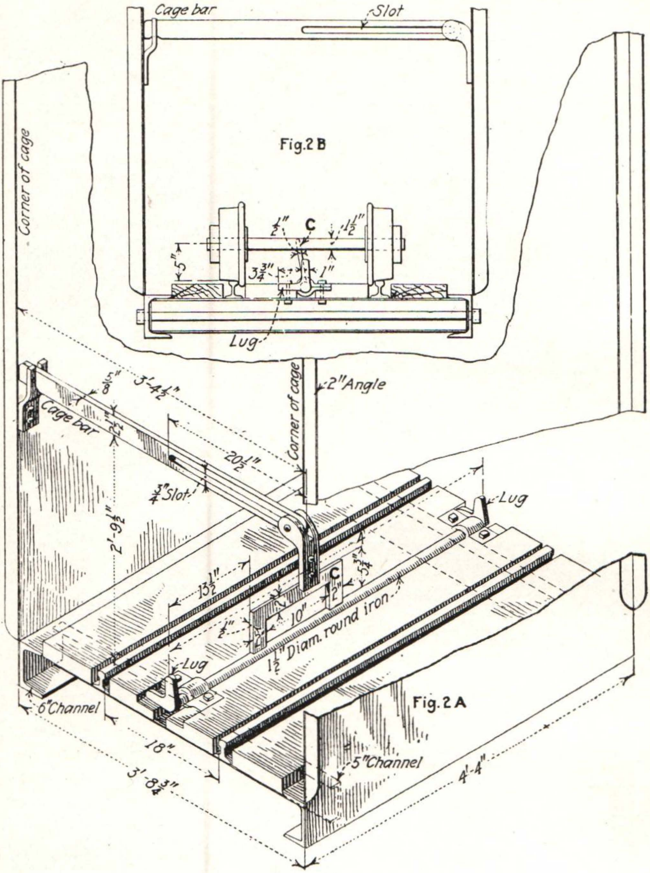
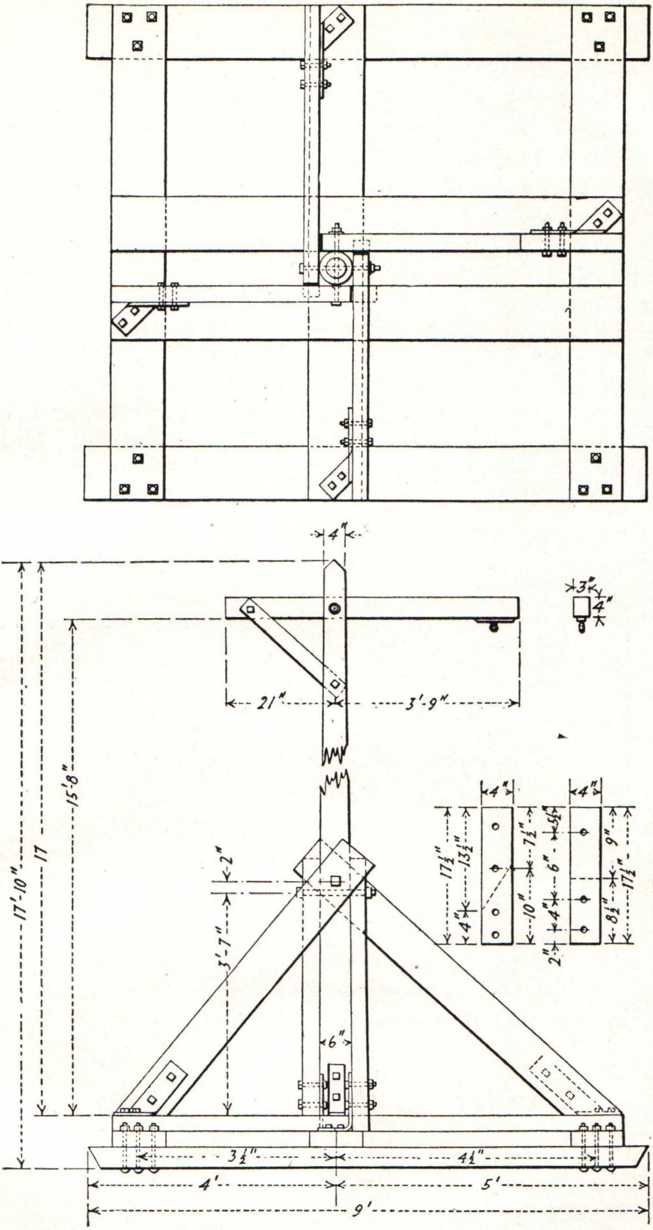


Fig. 2—Cage locking device and cage bars

in Fig. 1 prevents congestion of traffic by bringing in the empties from the same sides, at all times.

Portable Poles for Electrified Open-Pit Haulage

In the electrified open pit of the Hanna Ore Mining Co.'s Wabigon iron mine, on the Mesabi range, the locomotives take their current from an overhead wire, which is carried over the center of the track on the



Sketch showing design of portable pole or stand for carrying trolley wire at side of truck in electrified open pit main tracks and at the side on the loading tracks and on the dump. Above the main tracks the wire is suspended in the usual way and 22 ft. above the rail. This clearance is sufficient to permit a crane with its boom down to pass when an occasional derailment calls for its services. On the main tracks a pantograph collector is used. The center wire, however, would be in the way of the shovel boom if used on the loading tracks in front of the shovel. Likewise it would not be practicable on the dump, where the tracks are shifted from day to day. Temporary portable stands or poles are therefore used at the side of the track on the loading tracks and dump to carry the wire. This necessitates the use of a side-arm collector on the locomotive. The design of the portable stand is shown in the drawing.

The materials required for one of these stands are given in the accompanying table.

Hardware

No. of Pieces		
3	¾ in. x 16 in	Galvanized machine bolts
1	⅝ in. x 8 in	Galvanized machine bolts
12	½ in. x 4 in	Galvanized machine bolts
24	⅝ in. x 8 in	Galvanized carriage bolts
2	½ in .x 2½ in	Galvanized lag screws
2	½ in. x 3 in	Galvanized lag screws
4	$3\frac{1}{16}$ in. x 4 in. x 14 in	Galvanized bar iron
2	⅜ in. x 4 in. x 16 in	Galvanized bar iron
1	¼ in. x 1¼ in. x 28 in	Galvanized iron brace

Lumber

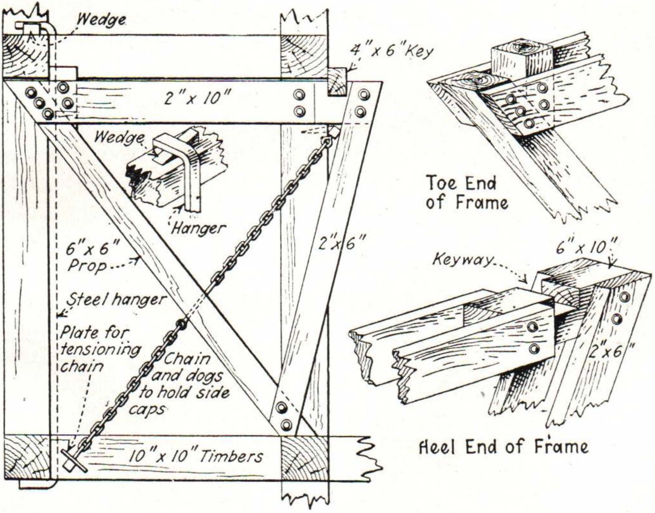
2 pieces	4 in. x 10 x 9 ft.	Norway pine
5 pieces	3 in. x 10 x 8 ft.	Norway pine
4 pieces	3 in. x 8 x 6 ft.	Norway pine
1 piece	3 in. x 4 x 5 ft. 6 in.	Norway pine

The portable stands are spaced about 40 ft. apart. The wire is carried 10 ft. from the center line of the track and 15 ft. above the rail. The design of the stands was worked out by the company's engineering staff and O. O. Ormond, superintendent.

Timber Frame for Underhand Square Setting

By George J. Young

Underhand stoping with square set timbering was described by Robert H. Dickson in the *Mining Journal-Press,* Jan. 6, 1923, p. 5. In this method of stoping the suspended square sets are supported by stulls and nail ing strips or by a boom with nailing strips to hold the lower members of the square set. In the accompanying sketch three separate devices are used for the purpose of holding the loose set and supporting the square set timbering above the undercut. The triangular-shaped frame carries the principal weight of the sets above the undercut set. Two frames are used; one on each side of the set. These are bolted together for convenience in getting them into the stopes. The lower side caps are held by a chain with a dog at each end for driving into the timbers. The dogs will hold in seasoned timber, but are apt to pull out of green timbers. A plate is attached to the lower dog so as to enable tension to be put upon the chain before driving the dog. Two chains with accessory dogs are required. The front lower cap is held by a hanger which consists of a light rail bent at each end. This is tightened in place by a wedge, as shown in the illustration. As soon as the lower set is under cut and the new set blocked, the triangular frames are removed and placed in position on the new set; the dogs are knocked out and the chains placed also on the new set. The hanger is then removed and similarly placed on the set below.



Details of support for underhand square sets

Notes on Tempering Drill Steel

By Letson Balliet

Mining Engineer, Tonopah, Nev.

THE BLACKSMITH SHOP is one of the most important "service" departments of a mine. If the blacksmith shop sends steel into the mine that breaks, batters, or is "out of gage," so that the longer drills will not follow the shorter into the hole, much time is lost by the drillers in struggling with poor steel. Thousands of dollars is lost in time wasted and in a curtailment of production. If the drill runners lose an hour or two each day "fighting steel," the muckers, the trammers, the hoist engineers, and other departments frequently lose time too. The service departments function only to facilitate production and development. The greater the production, the less the service departments cost per ton.

The mine blacksmith is not a steel metallurgist. He is a man who has learned by practical work and apprenticeship how to heat steel, to shape it, and to plunge it while hot into various liquors for hardening it. His operations are conducted by vision and "rule of thumb." When going from one mine to another he tries out several "tempers," as he calls it, until he gets the one that is best suited for the rock in the mine.

Troubles With "Mixed" Steel

The blacksmith frequently labors under one disadvantage: that is, a lot of mixed steel. Different salesmen call upon the purchasing agent and offer him steel at different prices. Price and delivery, plus salesman's statements, too often are the guide. If the buyer saves a cent a pound on a few tons of steel, the manager thinks he is doing well, but the result may be that the mine will have a mixed lot of steel. Sometimes six or eight different brands of steel are found in the same mine. An analysis would show carbon varying from 0.60 to 0.95, with varying amounts of silica, phosphorus, manganese, and other constituents. When the blacksmith undertakes to work such steel, if he treats all alike, forges at practically the same temperature, and tempers in the same way, he will find that the steels of different analysis have a different condition.

The standardizing of drill steel to analysis is more important than some mine managers realize.

It has been the practice of some manufacturers of ingot steel to cut off the top and bottom of the ingots and roll it into mining steel and to use the center of the ingot for uses where *good steel* is in demand. As a matter of fact, there is no use of steel that is as severe as mining. It must stand up under a continual vibration of the hardest kind and often with a lateral bending strain, combined with a rotating torque. No automobile axle, car axle, or moving parts of machinery puts a greater vibrating strain upon steel than an air drill.

When the blacksmith undertakes to sharpen steel he judges the temperature by his eye. On a cold, frosty morning, in a bright sunlight, on a cloudy afternoon, and in the shadows of the blakcsmith shop with the sun coming through openings at different angles, the appearance of the steel at the same temperature varies so much that mere observation is not an accurate way to gage temperature. Yet there is a critical temperature for each kind of steel.

CONTROLLING OXIDATION

The oxygen of the atmosphere, and of the air used in the forge or furnace, is severely oxidizing. Proper furnace conditions and accurate temperature control are necessary to reduce the bad effects of oxidization.

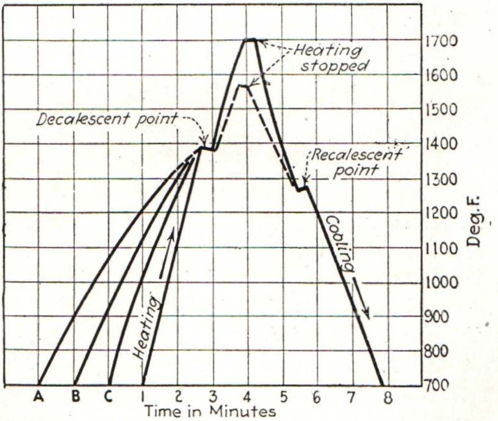


Fig. 1—Showing the critical temperature of heating

I will describe a furnace that can be installed at reasonable cost and that if properly used should lessen the losses from broken steel and time lost in underground operations. As compared with methods in use in some mines, it should mean that from 25 per cent to 40 per cent less steel for the same ore tonnage will come to the blacksmith shop for sharpening.

In Fig. 1 I have tried to illustrate the condition that exists when steel is heated. In a non-oxidizing (neutral) fire, the long "soaking" heat allows the molecules of the steel to assume a natural position that will avoid internal strains. A heat beginning at *A* and reaching the decalescent point in five minutes is better for the condition of the steel than one beginning at 1 and requiring but two minutes to reach the decalescent point. Even several minutes of annealing in the neutral fire below the decalescent point is beneficial and will tend to recondition steel that may have been previously heated or started to crystallize as a consequence of vibration.

CRITICAL RANGE OF TEMPERATURE

In the discussion of the proper forging and tempering temperatures, mention of the critical range or the critical point in steel frequently is made. A practical understanding of what is known as the critical range can be obtained by watching the slow heating of a piece of steel in the furnace. The steel brightens in color with the rising heat of the furnace until a point is reached where the steel apparently becomes a trifle darker in color than the furnace. If the heating continues, the steel again begins to increase in brightness, and once more asumes the same brilliancy as the furnace. The darkening is due to the absorption of heat at this point; and the temperature at which this absorption takes place is known as the decalescent or critical point. At this point the ferrite and pearlite are converted into austenite.

If the furnace is allowed to cool slowly, the point will be reached where the steel remains visibly brighter than the furnace, but in a few seconds it assumes the color of the furnace and darkens with it. This brightening of color is due to the throwing off of heat at the recalescent point where the austenite changes back to ferrite and pearlite. In mine steel there is a difference of about 100 deg. between the decalescent and recalescent points, and this 100 deg. is called the critical range. Obviously, tool steel, high-carbon steel, vanadium steel, and other alloys of steel have different points of decalescence and recalescence, although to the eye of the blacksmith the range of difference might be insufficient in varying lights for him to determine it. Accordingly, the pyrometer becomes absolutely necessary to gage furnace heat with accuracy.

If the heat is carried too high above the proper forging or tempering point, the molecules become too widely separated, and further heating above the critical temperature coarsens the grain. This coarse structure remains in the steel if it is allowed to cool undisturbed, but vigorous hammering or upsetting causes a coarse crystalline structure to be broken up and the grain refined. If, however, the hammering is stopped above the critical range, the coarse crystallization sets in again; and the higher the temperature above the critical range at which the forging stops, the coarser the structure will be. The finishing temperature for forging a drill-steel bit should therefore be at or slightly above the critical temperature, and the forging should be rapid and vigorous; it should not be accomplished by bulldozing or squeezing. If the hammering continues below the recalescent point, internal strain will be set up in the steel. To remove them it will be necessary to reheat and anneal the steel approximately to the decalescent point, so as to allow the molecules to assume their natural position without strain.

THEORY OF TEMPERING

The physical condition of the steel I have attempted to show in Fig. 2. The small sketch represents a bar of steel (idealized), of which the solid black end represents the steel at absolute zero with no space between the molecules. From this temperature various degrees of heat are represented until, at the other end of the

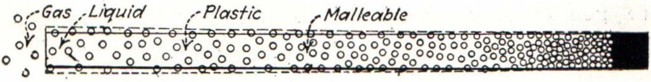


Fig. 2—Showing molecular structure at various temperatures

bar, the transformation to gas is indicated, wherein the molecules are so far apart as to have no cohesion. The only difference throughout the bar is the greater separation of the molecules where the heat is greater. Should a piece of steel in this condition have the hot end suddenly chilled by immersion in a cold liquid or solution, it is obvious that the molecules would immediately huddle together. The more violent the chilling, the quicker they would huddle together. If it were possible to have a piece of steel in this condition, ranging from absolute zero to say 5,000 deg., and suddenly immerse the whole piece in a quenching liquid, we would have an infinite number of temperature or molecular conditions throughout the length of the bar. There would be a different temperature at every point; the liquid in which it was immersed would be warmer than the cold end and colder than the hot end. Therefore, the molecules of the cold end would separate and those of the other end would huddle together. Some parts of the steel would be harder than others, some tougher, and some stronger, for it is obvious that not all points throughout the length of this bar would have the same strength, hardness, flexibility, and elasticity.

Within a narrowed temperature range, something similar in heat conditions prevails every time a mine blacksmith sharpens a piece of steel. He heats one end of the bar to a temperature of 1,500, 1,700, or even 1,800 deg., while he holds the cold end, in his hands. By conduction the heat is carried back into the steel for some distance.

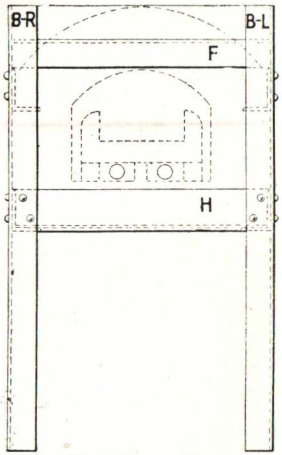
One of the most active, most damaging, yet most beneficial elements is oxygen. It is the one element the blacksmith must use, and yet, simultaneously, guard against. Iron or steel is just as much a *fuel* as is coal or wood. Steel will unite with oxygen and so will iron. The blacksmith frequently says, "I have burned the steel." That is exactly what he has done. Hot steel will also oxidize in water. Water generally contains entrapped air bubbles, and at high temperatures the oxygen from the water will assist in oxidation.

REVERBERATORY TYPE OF FURNACE

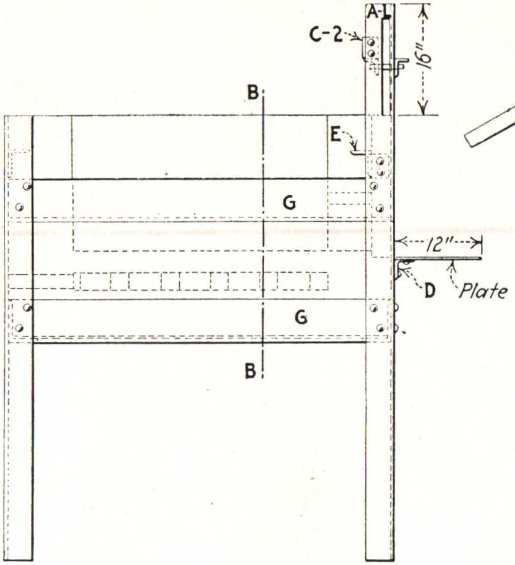
To some extent oxidation may be controlled by the furnace design. The drawing (on pages 50, 51, and 52) showing details of construction, illustrates a furnace of the reverberatory type, to be used with oil or gas fuel, wherein the flame does not impinge directly upon the steel and only the burned products of combustion can rise in the heating chamber where the oxygen has been greatly reduced and exists largely in the form of carbon dioxide or carbon monoxide. This special atmosphere may under some conditions have a tendency to cause the heated iron to take a little carbon out of the carbon monoxide. Thus case-hardening takes place. This is probably so small as to be unimportant, if indeed it is anything more than theoretical, but it at the same time prevents a part of the oxidation that would take place with the steel, and a possible burning of some of the carbon in the steel, as might happen with a severe oxidizing flame.

With some changes in the chamber it would also be possible to make this furnace for coal, coke, or charcoal, with the air draft coming in from below the grate bars and the burned gases of combustion rising around the heating chamber. This furnace can be built in any length or size that is desired. The main requirement is to make the combustion chamber the right size and to provide the right temperature in the heating chamber. This is regulated by a pyrometer. Any manufacturer of steel will furnish the analysis, the critical range, forging, tempering, and drawing temperatures for his product. This furnace can be made large enough for reconditioning the steel, including the shank; and it also may serve as an annealing furnace. The main object in the design is to keep the steel away from the blast of air through the fuel and avoid the possibility of the flame impinging directly upon the steel.

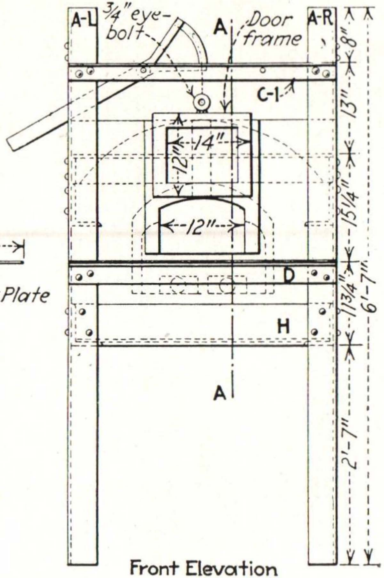
A question arises: "Why should a blacksmith temper steel?" Usually, the reason is that the blacksmith shop has a fire. It seems possible, particularly at larger mines, that economy might be effected if the blacksmith were permitted only to shape his steel, sharpen it, dress it, and send it to a specialist in charge of heat-treating and tempering. It would quite certainly increase the life of steel, decrease the amount of broken steel, and provide better steel for current use in the stope.



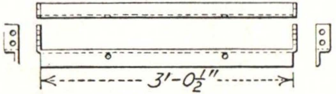
End Elevation



Side Elevation
Door and lever not shown



Front Elevation



Mark C-2

No.	Materials	Mark
2 Pcs.	4"x4"x1/2"x6'-7"L	A
2 "	4"x4"x1/2"x5'-3"L	B
1 "	2"x2 1/2"x3/8"x3'-1 1/2"L	C-1
1 "	2"x2 1/2"x3/8"x3'-7 1/2"L	C-2
1 "	3"x3"x1/2"x3'-1 1/2"L	D
1 "	3"x4"x1/2"x3'-5 1/2"L.	E
1 "	3"x4"x1/2"x2'-11 1/2"L.	F
4 "	4"x6"x1/2"x4'-5"L	G
2 "	4"x6"x1/2"x2'-11 1/2"L	H
1 "	1 1/2"x1 1/2"x1/4"x12"L for lever	
1 "	2"x2 1/2"x3/8"x4'-4"L for door	
1 "	1/2"x2"x3' W.I. for lever	
1 "	12"x3'-1 1/2"x1/4" plate	
50	3/4"x2" rivets	
6	3/8"x1" rivets	
150	-1 1/2" side arch brick	
500	Fire brick	
30	Fire brick splits	
4	12"x24"x3" fire tile	

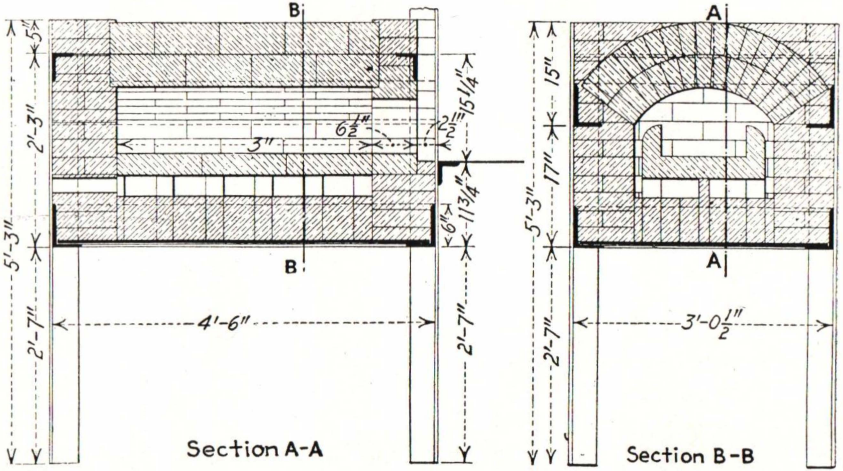


Fig. 2—Working drawing of a drill sharpening and tempering furnace

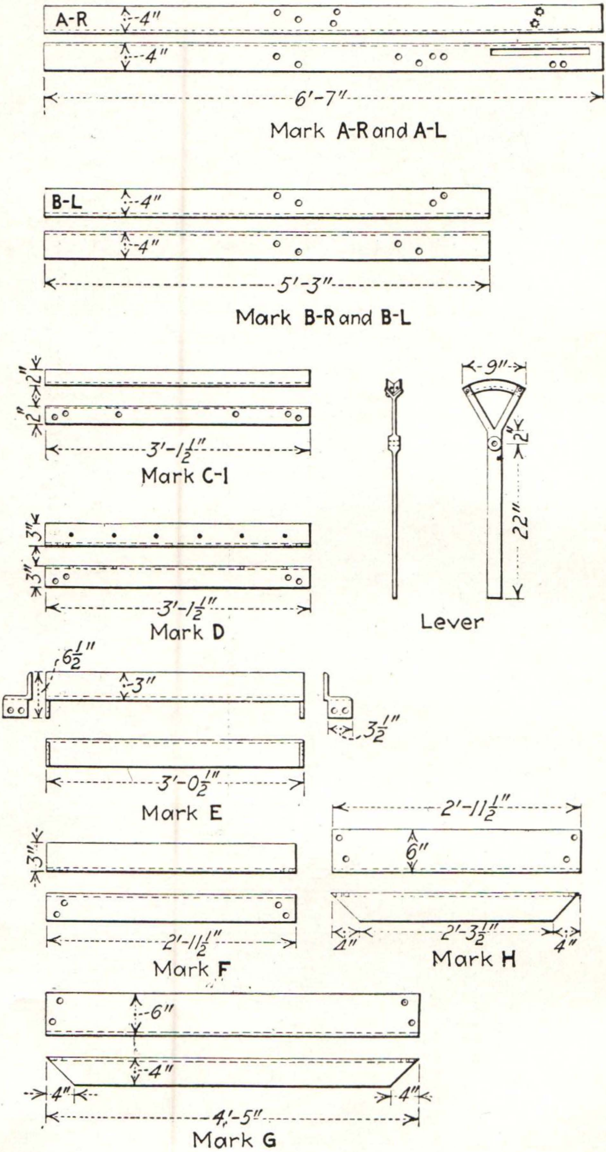


Fig. 2 (Continued)—Further details of furnace

PHYSICAL CHANGES OCCURRING IN THE FURNACE

Whenever fuel oil is sprayed, blown, atomized, or otherwise injected into a furnace or a hot combustion chamber—whether by steam or air, hot or cold—the physical properties of the fuel are unchanged, and although divided into the finest mist, it is still oil and not a gas. As such, each tiny globule must float in a bath of air (or oxygen) in order to burn. As each particle burns, the diameter of the globule diminishes, but the surrounding air is diluted by the products of combustion, and the oxygen content of the air reduced so that the rate of combustion decreases and the globule must float on and on, hunting more oxygen until it is all consumed or becomes chilled below the temperature of ignition.

Since oil cannot be burned without an excess of air, it is obvious that all oil fires are severely oxidizing fires. In coal and charcoal fires it is impossible to get complete combustion of the fuel without an excess of air; hence in the blacksmith forge or furnace there is not only a great likelihood that an excess of oxygen may be present, but it is extremely improbable that any blacksmith can handle his fire so as not to have an excess of air. Necessarily, oxidation is attacking the steel in the forge or furnace during the heating, and it is also attacking it when it is red hot and on the anvil.

ADVANTAGE OF GAS AS FUEL

Gas fuels can be made into completely combustible mixtures, wherein each and every molecule of the fuel-gas is in contact with the exact number of molecules of oxygen for complete union. Thus, a gas fire can be regulated to an oxidizing, reducing, or a neutral flame, whereas solid and liquid fuels can produce only oxydizing fires. Many shops at industrial plants have opportunity to use fuel-gases, but most mines use oil, coke, or coal in the blacksmith and tool-dressing shop.

The heating and tempering furnace in the accompanying drawing was designed as an oil-burning furnace, fed with oil and compressed air. Steam, low pressure fan air, or mechanical atomizers can be used in place of compressed air. Steam can be generated from a small water-tube boiler situated so as to use the stack heat. Likewise, the compressed air can be preheated by a coil in the stack before spraying the oil. For coke or coal fuel the design is not greatly different except that a grate or tuyère is substituted below the heating chamber.

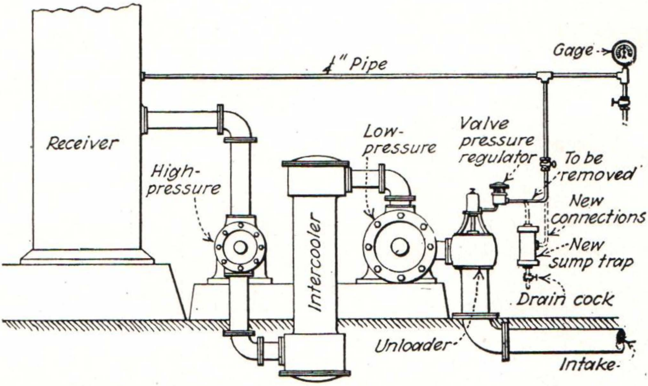
PROCESS INVOLVES APPLICATION OF A "SOAKING HEAT"

The purpose of this furnace is to prevent the flame and air blasts or air drafts from impinging directly upon the hot steel. The heating probably will be slower —what the blacksmith will call a "soaking heat." This is not a fault, but a desirable quality, as it allows the molecules of the steel to come to position slowly and homogeneously throughout, because of the constancy of the temperature through the entire piece of steel. The steel, no matter how hard, is tougher and less likely to crack or break. The furnace should be so large that it can contain so many pieces in the process of heating that the smith need lose no time, for while he is working one piece others can be in the fire. No time need be lost as the result of slow heating; in the end time is "gained." The steel "stands up" better, and for the same number of drilling machines the smith has to sharpen, say, only 50 per cent as many pieces for the same amount of underground work.

Overcoming Faulty Operation of Compressor Unloader

BY D. WELGE
Electrical Engineer, San Francisco

It is a common practice to connect the pressure regulator of a compressor unloader to an overhead pipe. The ¼-in. auxiliary pipe from the receiver to the pressure gage and pressure regulator as shown in the accompanying illustration was the arrangement in use at a mining property. The hot air carries considerable oil vapor and moisture, which condenses in the small pipe and flows by gravity to the pressure regulator, and fouls the delicate moving parts to such an extent as to make it inoperative. When the pressure regulator fails to function, the air pressure in the receiver either rises or drops, depending upon the position of the



Drain trap placed between receiver and unloader

unloader. If the unloader is held open, owing to this trouble, the air pressure will rise to a point where the safety valve will blow off. This blow-off pressure is set higher than normal working pressure, and therefore more power is taken by the electric motor or engine. It is not only a peak but a sustained peak for the total period of excessive air pressure.

Electric power schedules and power charges are usually based upon maximum demand kilowatt peaks. It therefore pays to give some attention to the pressure regulator of a compressor. If the position of the unloader is held closed, because of a faulty regulator, the air in the receiver will be consumed and the pressure will gradually drop.

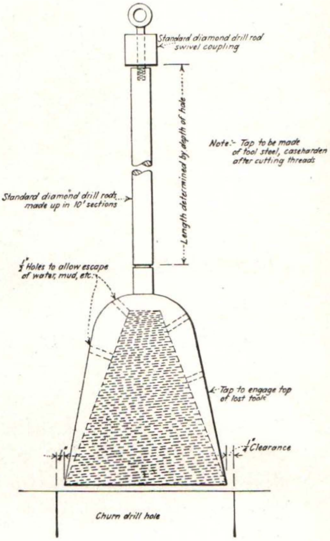
On a small Nevada mining property the compressor attendants found it necessary to dismantle the pressure regulator from two to three times each eight-hour shift. Small quantities of oil, water, and carbonized oil specks would stick the moving parts together so tightly that considerable force was necessary to get them apart for cleaning. This trouble was overcome by installing a trap as shown by dotted lines in the illustration. The trap can be made from standard pipe and fittings—2½-in. diameter pipe by 12 in. long with reducers at top and bottom. It is important to place the trap below the pressure regulator and to open the drain cock once each shift to blow out its contents. This is both quicker and cheaper than having to dismantle the regulator at intervals of three or four hours.

Recovering Churn-Drill Tools

BY ROY H. POSTON

St. Francois, Missouri

Churn-drill tools often become loosened at the joints or are broken off at some point and fall back into the hole while the drill is in operation. The subsequent task of recovering the lost section is not always easy. Such a condition was recently obtained at a southeast Missouri mine when the entire string of tools, weighing 1,468 lb., together with 60 ft. of 2¼-in. manila rope, broke off the main cable while drilling and fell to the bottom of a 225-ft. hole. The force of the fall caused the top of the tools to become embedded in the side of the wall, and this position further complicated their recovery.


Standard diamond drill rod swivel coupling
Length determined by depth of hole
Note:- Tap to be made of tool steel, caseharden after cutting threads
Standard diamond drill rods made up in 10' sections
½" Holes to allow escape of water, mud, etc.
Tap to engage top of lost tools
¼"
¼" Clearance
Churn drill hole

Special tap used in the recovery of churn-drill tools

Some ten days were consumed in cutting and removing the rope and in trying to get over the end of the tools with standard churn-drill fishing tools. A tap, used in conjunction with standard diamond-drill rods finally accomplished the desired result, and the lost tools were retrieved in an hour and forty minutes.

In operation the tap was fastened to the diamond-drill rods and lowered into the hole by the addition of other rods until the tap rested on the upper end of the churn-drill tools. Hand pipe wrenches were then employed to turn the rods, until the tap threads began to engage the top of the rope socket. Additional leverage, by means of pipes over the end of the wrench handles, forced the tap to take firm hold of the rope socket. When this had been accomplished it was only necessary to connect the diamond-drill rod swivel to the sand line and pull the tools loose and out of the hole.

Cushioning an Ore Feeder Arm

By Charles Labbe

In certain types of ore feeders, the motion to the arm or tappet operating the feeder is given by the striking of a blow upon it by the stamp or other actuating mechanism. The stiff connection provided and shown in Fig. 1 is not well designed to receive the blow and can be readily improved as shown in Fig. 2.

In this design the feeder arm is reamed out to receive a loosely fitting sleeve which encircles the bumper rod. A spring is interposed between the upper adjusting nut and the top of the arm. This should be stiff enough to transmit the blow but elastic enough to absorb the shock. The adjustability of the bumper rod is not interfered with.

The arrangement shown will reduce the wear upon all of the parts and prolong the service of the parts subjected to direct impact.

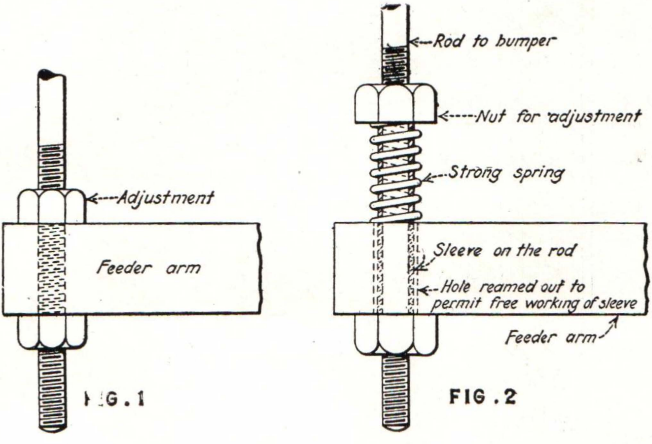
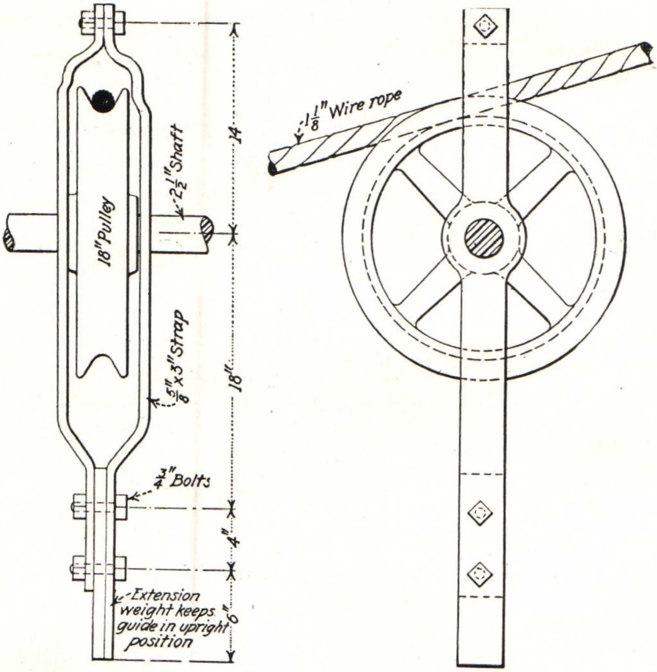


Fig. 1—Ore feeder arm of rigid type. Fig. 2—Improved ore feeder arm designed to avoid shock

Rope Guide for Idler Sheaves

BY L. C. MOORE
Hibbing, Minn.

A device for preventing hoisting ropes from pumping off idler sheaves was successfully used at the Stevenson mine of the McKinney Steel Co., on the Mesabi Range. The guide is constructed of two $\frac{5}{8}$ by 3-in. iron flats, which are bored to pass over the shaft as shown in the accompanying sketch and are shaped to give a small clearance to the upper rim of the sheave. The side pieces are extended so as to bring the center of gravity of the guide below the center of the shaft. There is plenty of room for the hoisting rope to play, but the guide prevents the rope from losing the groove of the sheave. Very little wear results, as the guide is moved along the shaft by the hub of the sheave.



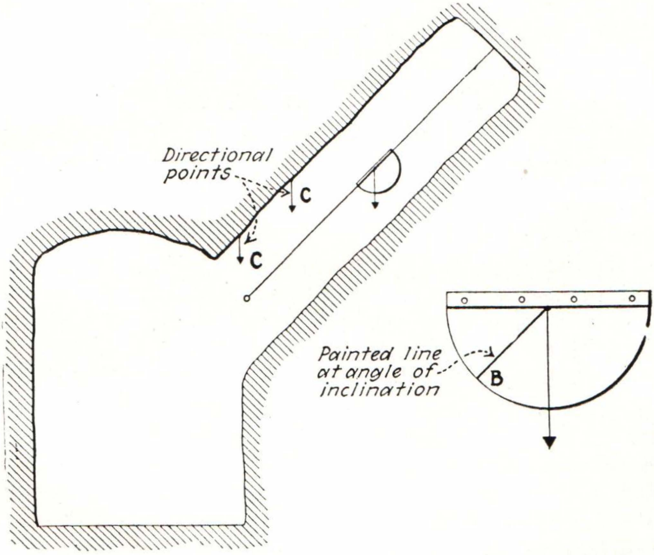
Guide for idler sheaves, in upright position

Aligning Raises

By C. A. Fredell
Apartado 79, Pachuca, Mexico

Several methods for the alignment of raises have been described in *Engineering and Mining Journal* and generally one method is as good as another, provided proper precautions are taken in applying it. However, some of the methods are not advantageous under all conditions, more especially where the raise is a single compartment and untimbered. The following method, on the other hand, has been successfully used under varying conditions, and has been found to be suitable for single-compartment raises where sighting boards are likely to be knocked out:

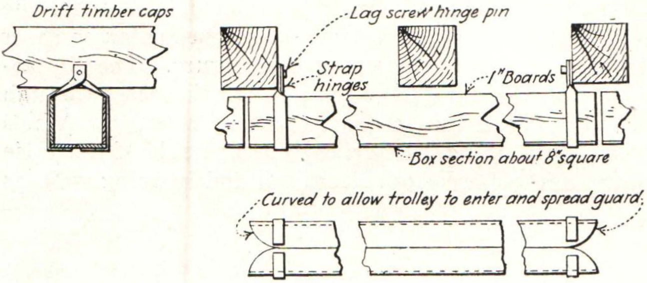
Having determined the inclination of the raise, the angle is laid out upon a clinometer constructed of sheet tin as shown in the accompanying figure. The clinometer is provided with a stout cord which passes through several holes in the base line of the clinometer. A hole 12 to 16 in. in depth is drilled into the side of the raise at the mid-point between footwall and hanging wall, as



A simple method of aligning a raise

at *A* in the figure. By tying one end of the cord to a drill steel or tamping stick placed in the hole and carrying the other end of the cord to the face of the raise, the approximate center line is quickly ascertained by raising or lowering the cord until the plumb bob hangs directly over the painted line at *B*.

In a long raise the point *A* is advanced every 50 to 60 ft. The inclination of the raise can be changed at any of these points by removing the painted line and placing a new line upon the clinometer. Direction points are set in plugs in the hanging as at *CC* in the figure.


Drift timber caps
Lag screw hinge pin
Strap hinges
1" Boards
Box section about 8" square
Curved to allow trolley to enter and spread guard

Trolley Wire Guard

BY E. H. ONEAL

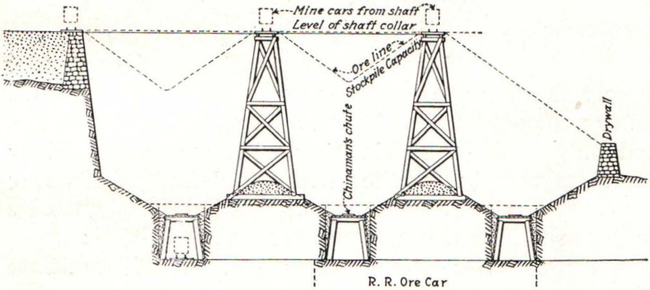
Mechanical Engineer, Old Dominion Mine, Globe, Ariz.

A new type of trolley wire guard is being successfully tried out at the Old Dominion Mine, Globe, Ariz. It is of the box type, closing in the trolley wire at the sides and bottom, as shown in the accompanying drawing. The box is constructed of 1-in. boards, which are fastened to hangers or hinges as shown. The length of each section and the clearance dimensions depend upon the location, whether on curves or tangents, and whether the trolley wire is directly over the track or at one side. Each section is so cut at either end as to allow the trolley to enter and leave the guard, pushing the two portions of the box outwards. The guard was designed and built by W. J. McCune, framing shed foreman for the Old Dominion company.

Reclaiming a Stockpile by Chinaman Chutes

BY W. W. RAYMOND
Arizpe, Sonora, Mexico

A modification of the Chinaman chute was used at a mine in Arizona in reclaiming a stockpile and loading it into railroad cars. As is often the case the shaft was located on a hillside higher than the loading siding. Along the railway siding an excavation was made with its floor 10 ft. above the height of an ore gondola and extended back from the track about 60 ft. where a dry retaining wall was built up to the level of the shaft collar which was about 30 ft. above the cut floor.



Design of stockpile arrangement for hand loading and tramming

Three trenches, 8 ft. wide and 11 ft. deep, as shown in the accompanying figure, were dug in the floor of the cut at right angles to the siding and 30 ft. center to center. Drift sets were erected in the trenches and lagged on the sides and top, leaving 18-in. spaces between the lagging over the center of the caps, and 2x8-in. plank, 2 ft. long, was placed bridging this opening, cleats 17 in. outside to outside being nailed on the planks to engage the lagging. The space between the lagging and the sides of the trench was filled and the spoil from the trenches was trimmed and tamped to form a ridge between trenches with sides sloping 30 deg. Trestles were erected over the center of the ridges to carry the tracks from the shaft.

The stockpile was accumulated by dumping mine cars from the trestles. When a shipment was made the gondola was spotted with its down-grade end opposite one of the drifts and a length of track, which bridged the gondola from side to side, was connected with the track under the Chinaman chute. A mine car was spotted under the front toe of the stockpile and the chute was operated by prying out the outside 2-ft. plank. A forty-ton car could be loaded in eight hours by one man with another to trim and help move the gondola.

How to Clean the Oil Reservoir of a Gasoline Engine

By G. S. Hamilton
Climax Engineering Co., Clinton, Iowa

Once every year it is desirable to drop the oil pan and thoroughly clean the reservoir of a gasoline engine. Simply changing the oil in the crankcase is insufficient. After a certain length of time, sediment will accumulate, and only a thorough cleaning will remove it. Many operators may be familiar with the details of cleaning the reservoir, but for the new user of industrial gasoline engines the following instructions will be helpful:

1. Clean outside pan to keep grit out of crankcase when open.
2. Drain off old oil.
3. Remove the oil-pan stud screws and bolts.
4. Drop pan being careful not to damage gasket.
5. Wash out oil pan with kerosene or gasoline and dry out with lintless cloth. Do not use waste. The threads and lint are likely to clog the oil duct.
6. See that crankcase gasket is in good condition, and in proper position. If necessary, replace with a new one.
7. Cement gasket tightly to oil-pan flange, using shellac.
8. Cover upper face of gasket with grease or hard soap soaked in water. This will check leakage and permit oil pan to be removed when necessary without breaking the gasket.
9. Replace pan and tighten screws and bolts tightening a little at a time on each one to distribute the strain evenly.
10. Refill oil reservoir with proper amount of fresh oil of correct grade and see that indicator registers "full."

Deepening the Pilares Shaft of Moctezuma Copper Company

By E. Leland
Mine Superintendent, Moctezuma Copper Co.,
Nacozari, Sonora, Mexico

THE NEW Pilares shaft of the Moctezuma Copper Co., Pilares de Nacozari, Sonora, Mexico, is approximately 300 ft. outside the mineralized ore zone and is in latite breccia to a point below the 700 level. From this point down it is in andesite and andesite breccia, a solid uniform rock, but very blocky in places, so that when left unsupported it sloughs badly. This shaft now extends to the 1,800 level. The last 400 ft., from the 1,400 to 1,800 level, has just been completed.

The Guadalupe shaft, near the center of the mineralized area, is the main hoisting and drainage shaft and has always been the pilot shaft. As each new level was opened up, main drifts were driven from the Guadalupe to the Pilares shaft and pilot raises driven from level to level. Formerly, after the pilot raise was driven, enlarging and timbering was started at the bottom and a lined chute carried in one of the shaft compartments for waste disposal. Some of the disadvantages of this method were the difficulty in handling the timber, the extra cost of the lined chute, and the necessity of breaking the rock rather fine for handling through the chute. This shaft was sunk by the method mentioned from the surface to the 1,400 level.

Early in 1924, when it was decided to extend this shaft to the 1,800 level, the following plan was adopted: A pilot raise, Fig. 1, 4x7 ft. in section, was driven as closely as possible in the center of the shaft location from the 1,800 and from the 1,600 levels, to the bottom of the sump 35 ft. below the 1,400 level. The 1,800 raise connected with the drift on the 1,700 level, continuing on to the 1,600 level. The raise from the 1,600 level connected with the sump below the 1,400 level, a distance of 167 ft.

On the completion of the pilot raise, a shrinkage stope was started 20 ft. above the 1,600 level, as near 12x20 ft. in section as possible, to accommodate the shaft timbers, which require an over-all space of 11x19 ft. A shrinkage stope shovelway, Fig. 2, was built

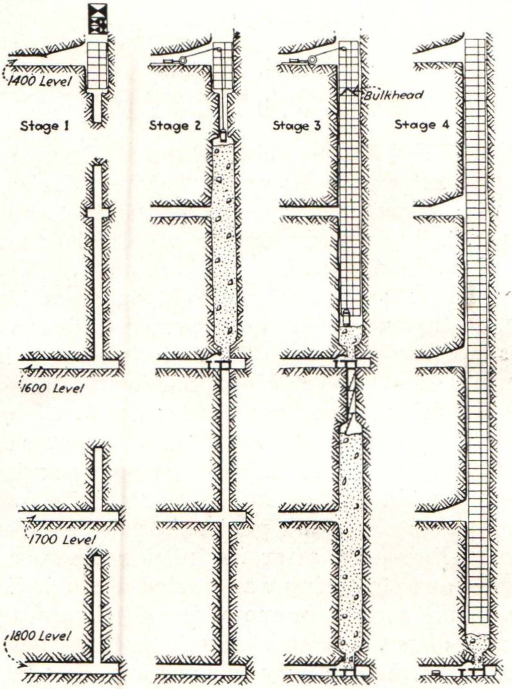


Fig. 1—Sketch showing various stages of Pilares shaft sinking

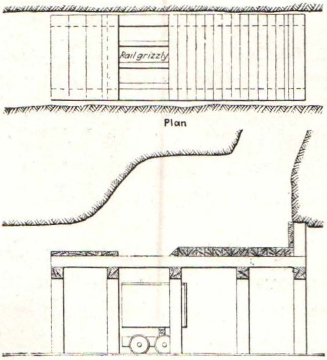


Fig. 2 — Loading platform at each level

on the 1,600 level so arranged that the raise from below was left open and protected. All the over-break was drawn off through the "Chinaman chute" shovelway as the shrinkage operation progressed. The same procedure was carried on with the section between the 1,800 and 1,600 levels.

After breaking through with the shrinkage into the bottom of the shaft, and prior to starting of timbering, it was necessary to protect the workmen from possible accidents and falling material in the operating section of the shaft between the surface and the 1,400 level. A solid bulkhead was placed just below the 1,400 level in the main hoisting compartment, *A*, Fig. 3, the counterweight compartment, *D*, and hoisting compartment, *C*, to allow for sufficient clearance below the cages in compartments *A* and *C* and the counterweight in *D*. A bulkhead was placed just above the level in Compartment *E* to allow free access by means of ladders to the work below. In Compartment *B* a bulkhead was placed 50 ft. above the 1,400 level and cage service in this compartment was stopped at the 1,300 level. Service in compartments A and C continued to the 1,400

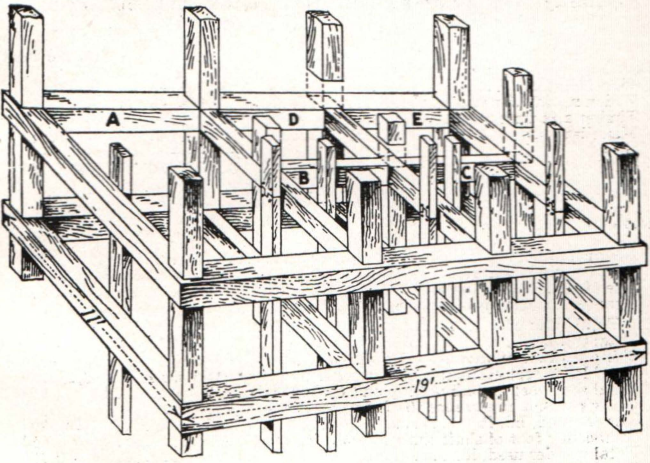


Fig. 3—Perspective sketch showing shaft timbers

level throughout the deepening process from the 1,400 to 1,800 level.

A small service hoist was installed at the 1,400-ft. station and service maintained in Compartment B, through which all timber and supplies were handled for the job. The shaft was timbered throughout with redwood. The wall plates, end plates, and posts are 12x12 in.; dividers are 10x12 in., and the 8x12-in. lagging is 2-in. redwood plank.

As each set was hung the broken rock was drawn down sufficiently for the placing of an additional set. In some places the walls had sloughed to an extent that bridging behind the sets was necessary. However, little trouble was experienced on account of over-break during the shrinkage operation, and because of the careful engineering supervision no extra rock had to be cut to make room for the timbers.

The operations incident to the deepening of this shaft may be divided into: (1) The driving of the pilot raises; (2) the shrinkage of the shaft to full size required for timbering; and finally, (3) the placing of the timber.

The following cost and efficiency data are submitted:

Sinking New Pilares Shaft from 1,400 to 1,800 Levels

Summary	Amount	Cost per Foot
Pilot raises	$2,681.47	$7.066
Widening and shrinkage	2,846.46	7.501
Timber and timbering	15,998.29	42.156
Miscellaneous	4,200.48	11.068
	$25.726.70	$67.791

Statistics

Distance sunk, ft.	379.5
Distance timbered, ft.	382.5
Size of shaft excavations, ft.	12 x 20
Over-all dimensions of timber, ft.	11 x 19
Total labor cost	$9,111.65
Total shifts	2,568½
Shifts per foot of advance (*sinking only*)	6.76
Total labor cost excluding shop expense and timber framing	7,272.99
Total shift cost excluding shop and timber framing	1,954½
Shifts per foot of advance (*timbering* and *framing*)	5.15
Timber used, bd. ft.	248,909
Timber per foot of shaft timbered, bd. ft.	650
Total powder used, lb.	3,778
Powder per foot of shaft, lb.	9.95

Pilot raises started March 8, 1924.
Shaft in operation Dec. 15, 1925.

Details and Distribution of Costs

Pilot Raises		Amount	Cost per Foot
Labor			
Breaking		$844.85	$2.449
Mucking		231.15	.670
Supplies		29.00	.084
Drills and tools		802.40	2.325
Explosives, 1,716 lb.		660.54	1.915
Timber, 2,640 bd. ft.		113.53	.329
Total		$2,681.47	$7.772
Feet of raising			345
Feet per shift—contractor breaking only			2.09
Pounds powder, per foot			4.973
Contract breaking shifts	165		
Contract mucking shifts	186		
Total shifts	351		
Feet per shift			0.983

Widening and Shrinking	Shifts	Amount	Cost per Cubic Meter (35.3 Cu.Ft.)
Labor			
Shaft	277½	$1,831.07	$0.774
Shop	10	30.35	.013
Drills and tools		553.03	.234
Explosives and handling		432.01	.183
Total		$2,846.46	$1.204
Tramming cost 15c. gold per ton-car not included in this cost			
Pounds of powder used	1.632 per cubic meter		$0.6899
Volume removed	83.502 cubic meters		2.364
Distance shrunk	379.5 ft.		

Timbering	Shifts	Amount	Cost per Ft.
Labor			
Framing	466¾	$1,428.30	$3.734
Placing	861	3,160.94	8.264
Shop	38	249.10	.651
Total		$4,838.34	$12.649
Timber and supplies, 212,346 bd. ft.		11,119.95	29.072
Incline charges		40.00	.105
Total timbering cost		$15,998.29	$41.826

Labor represents 382.5 ft. of completed work.

Miscellaneous Expenses Incident to Completing Shaft

Labor	Shifts	Amount	Total
Sundry contracts	465	$1,204.98	
Shops	59⅝	130.91	
	524⅝		$1,335.89
Supplies			1,029.22
Drills and tools			111.49
Explosives and handling (431 lb.)			114.23
Timber, 33,923 bd. ft.			1,609.65
			$4,200.48

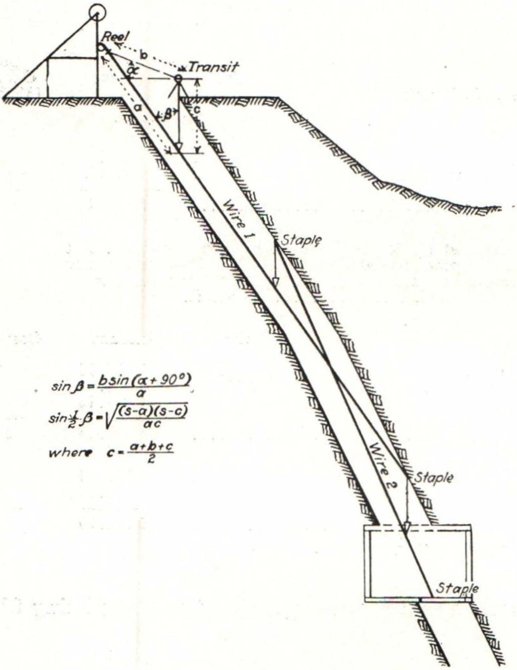
Including building bulkheads, safety devices, installing equipment, and incidental supplies and labor.

All amounts and costs are calculated in United States gold.

Surveying Steep Inclined Shafts Without an Auxiliary Telescope

By Edward H. Wisser

It is occasionally desirable at a small property with limited engineering equipment to carry a transit survey of reasonable accuracy down a steeply inclined shaft. This problem arose at a gold property in the Southwest, developed through three inclined shafts, 4x7 ft. in the clear, about 250 ft. apart, and each about 200 ft. deep.



Reel
b
Transit
α
a
β
c
Wire 1
Staple
Wire 2
Staple
Staple
$\sin\beta = \frac{b\sin(\alpha + 90^\circ)}{a}$
$\sin\frac{1}{2}\beta = \sqrt{\frac{(s-a)(s-c)}{ac}}$
where $c = \frac{a+b+c}{2}$

Sketch showing method of making survey

The dip of the vein and the inclination of the shafts averaged about 65 deg.; two of the shafts were connected by drifts. The transit in use was of the "light mountain" variety, and had no auxiliary telescope. A complete survey of the underground workings was wanted immediately, as no reliable mine maps were in existence.

The method adopted was that of stringing No. 24 (B. & S.) iron wire down the shafts, setting up at the collar over a point on the wire and transferring the azimuth from the surface traverse to the wire. Set-ups were made at the various levels, and the azimuth was transferred to the underground traverse stations. The inclination of the wire was obtained by two methods:

1. By measuring the vertical angle to a point on the wire, up the wire from the transit, and taping the distance along the wire from the point vertically beneath the transit to the point sighted, together with the distance from the horizontal axis of the transit to the point vertically beneath the transit (the "H. I."). This gave two sides and the angle opposite one of them of an oblique triangle.

2. By tapping all three sides of the above triangle.

The inclination of the wire was measured in this manner at the collar and at each level, and the resulting values were averaged. The greatest single variation at any of the shafts was 12 minutes. (The vertical circle of the transit read to 5 minutes only.) The distance along the wire from the point at the collar to the points at the different levels was taped; spads were put in above these points by plumbing. A convenient reel was improvised for stretching the wire and placed above the collar, generally on the headframe. The wire was fastened to a staple near the bottom of each shaft, and the maximum safe tension applied.

The chief difficulty encountered was in setting up in such a manner as to get the greatest possible distance from the transit to the point on the wire used in measuring the vertical angle; in this case, it was just possible to get this point within the zone of focus of the transit and at the same time low enough down the wire so that the point could be sighted. The maximum angle that could be read was about 50 deg.

A further difficulty was met in the main working shaft, caused by the fact that the shaft steepened about 10 deg. about halfway down. It was not feasible to set up at this bend, because it was impossible, owing to the smallness of the shaft, to sight to a point down the wire from the transit. To overcome this, a second wire was strung down the steeper part of the shaft and fastened to a staple exactly above the first wire. This wire was lined up by moving the end, near the bottom of the shaft, until the wire came exactly beneath a plumb bob dropped from the end of the first wire, as shown in sketch. The azimuth was transferred and the two vertical angles were measured as described above; the distance from the point at the collar to the intersection of the two wires, and from the latter to the point at the bottom, was measured.

A closed traverse was run between the two connected shafts; the horizontal error of closure indicated an accuracy of 1: 4000; the azimuth checked to 1 min. 30 sec.; elevations to 0.08 ft. This was sufficiently accurate for the purpose required.

Underground Ore-Bin Construction at the United Verde

By C. E. Mills

Chief Engineer, United Verde Copper Co., Jerome, Ariz.

At the United Verde mine, Jerome, Ariz, ore from below the Hopewell tunnel, which is at the 1,000-ft. level, is delivered by skips to two bins which discharge through gates into standard-gage cars hauled by elec-

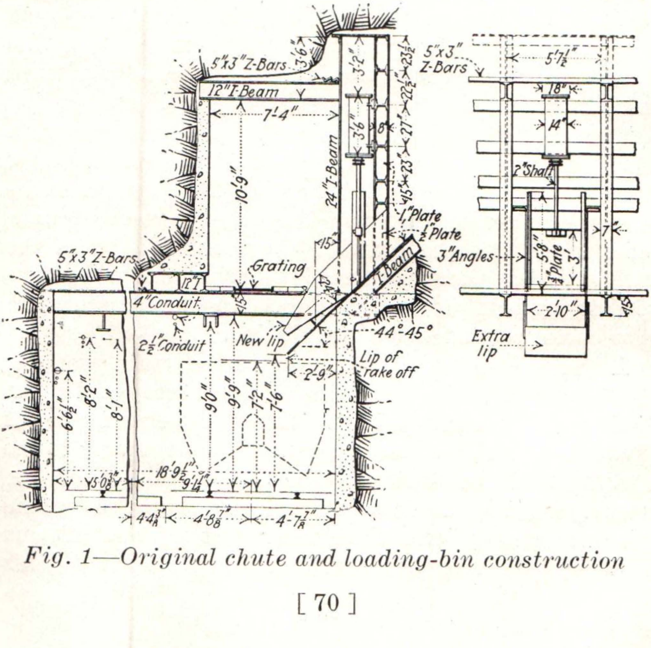


Fig. 1—Original chute and loading-bin construction

tric locomotive on the tunnel level. Loading chutes and bin construction are shown in Fig. 1. The front support of the bin is formed by six vertical 24-in. I-beams which rest on a concrete wall and are braced by 12-in. I-beams near the top and 15-in. I-beams at the bottom, and the front wall of the bin is formed by horizontal H-beams bearing against the vertical I-beams and supporting steel plates. The bottom of the bin consists of 1-in. plates supported by 12-in. I-beams, inclined at an angle of 45 deg. There are five chute gates to each bin. The vertical I-beams are at 5 ft. 7½-in. centers. Gate opening is 2 ft. 10 in. wide and the gate of ¾-in. plate is 3 ft. high. The gate slides in grooves formed by a pair of angles. It is operated by a 14-in. O.D. air cylinder. The clear opening of the gate is 2 ft. high. This chute and gate have proved satisfactory for mine ore, which must first pass a grizzly, the openings of which are 12 by 15 in.

An improved chute and gate construction was designed for the transfers delivering ore from the open pit. All ore is delivered over grizzlies on the 900 level, the bar spacing of which is 15 in. Ore is received in a vertical transfer (8 by 15 ft. in cross-section) which terminates in an incline leading to a pair of gates and chutes. The gate opening is 5 ft. wide and 2 ft. 3 in. high. The smaller gate opening used in the shaft ore bins proved to be too small for steam-shovel ore, which is occasionally wet and sticky and contains more or less old timber. A standard bin for all loading bins on the 1,000 level is shown in Fig. 2. It is simpler in construction than that shown in Fig. 1. As will be noted, the bin front is formed by three vertical 18-in. I-beams which are anchored at the bottom to a concrete wall forming one side of the tunnel and concreted in a hitch at the top. Three 15-in. I-beams are used to brace the lower ends of the vertical I-beams; and the ends of these beams are placed in hitches and concreted in place. The front wall of the bin is formed of vertical rails supported by 8-in. I-beams. The sloping floor of the bin is constructed of 40-lb. rails 10 ft. in length. These rails are riveted in groups of three to Z-bar clips and are concreted in place. Guides for the gates are angles 3 by 5 in. by ⅜ in., 6 ft. 4 in. in length. Gates are constructed of ¾-in. plate stiffened by three lengths of 60-lb. rail.

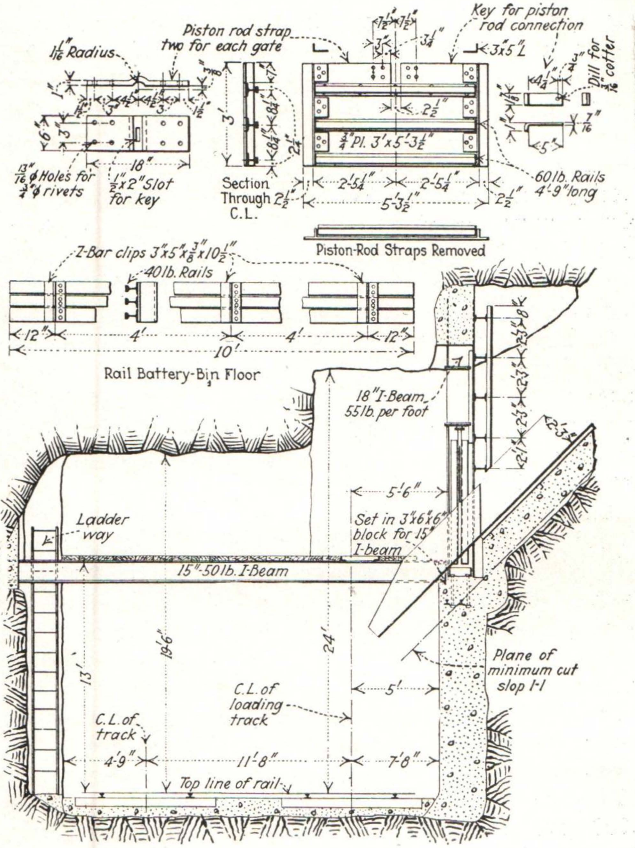


Fig. 2—Improved chute and loading-bin construction

Details of the chute are given in Fig. 3. An operating platform is supported by the horizontal I-beam braces and access is provided by an arched crosscut and offset ladderway as shown in Fig. 2.

The design shown in Fig. 2 has proved to be sufficiently rugged for the service, and few repairs have been necessary since the first installation in 1924. Loading time for a 10-car train of 30 tons per car averages 5 minutes, which represents 8 per cent of the total time

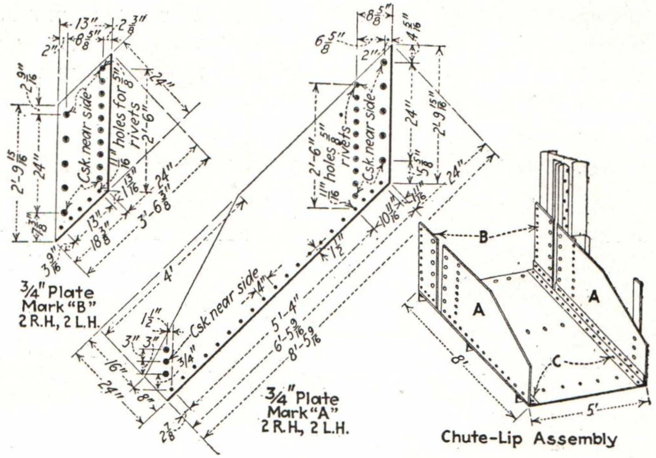


Fig. 3—Details of chute lip

per trip on a 1.5-mile run to the transfer bins at the portal of the Hopewell tunnel.

Cost Sheet for the Mine Foreman

FROM AN ARTICLE BY WILLIAM F. BOERICKE

The cost sheet for the foreman should emphasize, and give detail to, those items which come directly under his eye, and which by his efforts to improve will bring the greatest saving to the company. Everything else should be subordinated.

A practical form for his use is suggested here. This form of cost sheet gives everything essential for the foreman, without distracting his attention to operations over which he has no control. Emphasis is placed on subdivision of labor, and this may be gone into in greater detail if desired. The "B" division of material and supplies may be still further compressed if some of the items amount to little. Under this head comes the question of including the cost of new equipment, such as a new pump or a new drill. Common sense indicates that the cost of such items, if they are to be included in the cost sheet and not charged directly to capital account, and subsequently amortized, should be distributed over several months' time, so that the cost in any one month should not appear disproportionately large.

Mining Cost Sheet for ...... Mine Month of 192

A. Labor Cost	**Total Cost**	**Cost Per Ton**	**Year's Average**
Shovelers	......	......	......
Drillers	......	......	......
Tr-Trammers	......	......	......
Timbermen	......	......	......
Powdermen	......	......	......
Trackmen	......	......	......
Pumpmen	......	......	......
Shaftmen (landers, cagers, skipmen—not men in shaftwork, who should be charged to development)	......	......	......
Other miners	......	......	......
Total labor cost	......	......	......
B. Material and Supplies			
Powder, including caps and fuse	......	......	......
Timber	......	......	......
Drill repairs and supplies	......	......	......
Track supplies	......	......	......
Other mine supplies	......	......	......
Total material and supply cost	......	......	......
C. Power			
Kw.-hr., compressor, at .. cents per kw.-hr.	......	......	......
Kw.-hr., pump, at .. cents per kw.-hr.	......	......	......
Kw.-hr., miscellaneous underground use, not including hoist	......	......	......
Total power cost	......	......	......
Total mining cost			
Tons hoisted			
Concentrates produced			
Per cent recovery			

Straightening and Bending Devices

A simple device, designed and constructed in the shops of the Argonaut Mining Co., at Jackson, Calif., and used for straightening stamp stems or for bending pipe, is shown in Fig. 1. The base of an old mortar box, cut down to a convenient height, affords a solid table, and the bolt holes enable the pressure screw to be shifted where convenient. A cast-iron screw block is bolted to two short lengths of 6-in. angles, and these in turn are held down by four bolts passing through short lengths of pipe as shown in the sketch. V-blocks are used to support the object to be either bent or straightened. A cap is placed on the end of the screw,

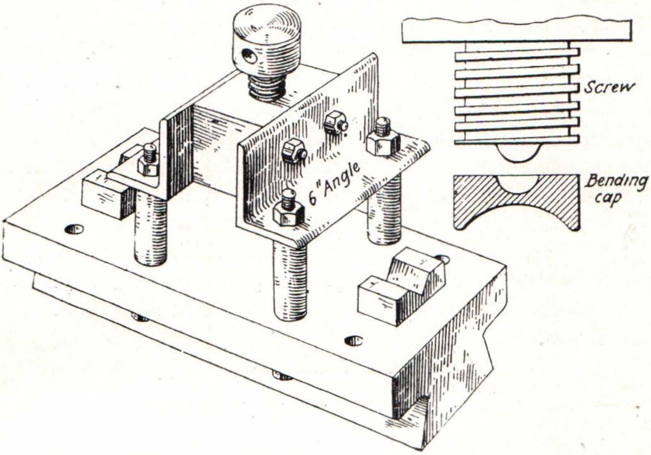


Fig. 1—Shop rig for straightening and bending pipe or rods

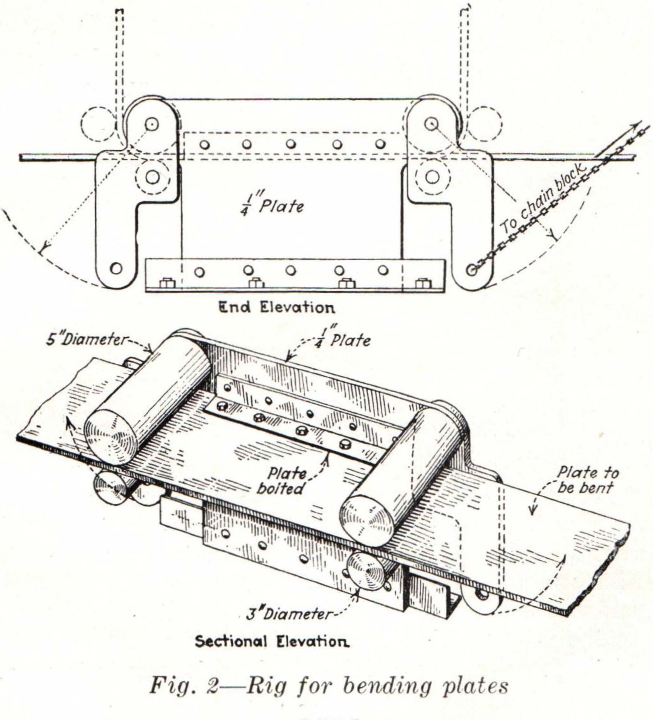
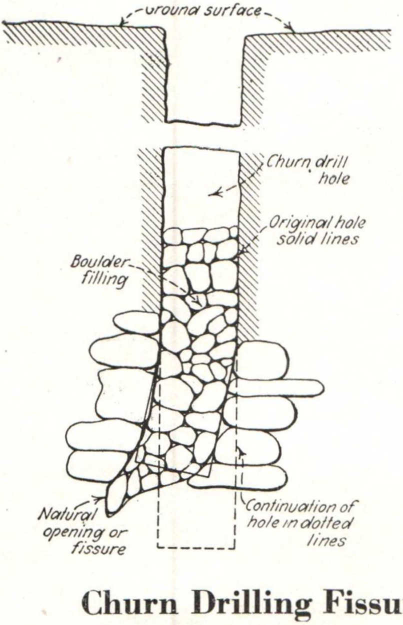


Fig. 2—Rig for bending plates

which protects the surface from injury and gives a firm bearing for the screw.

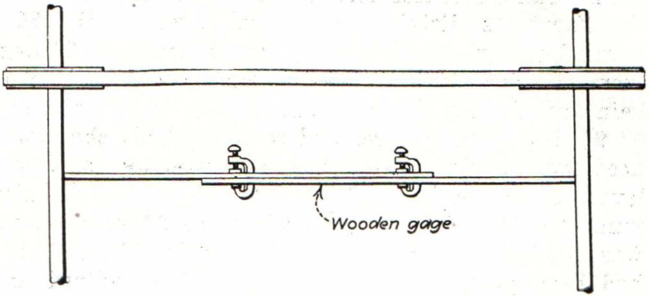
For bending plates used in the repair of mine cars, two pairs of bending rolls are shown in Fig. 2. The plate to be bent, after being cut to dimensions and punched, is slipped between the rollers and bolted to angles on either side of the bending rig, as shown in the sectional view. The lower rollers, which are carried in a frame swinging concentric with the large stationary roller, are drawn up by a chain block, bending the ends of the plate to an upright position. By removing the bolts the bent plate can then be lowered and removed.


Ground surface
Churn drill hole
Original hole solid lines
Boulder filling
Natural opening or fissure
Continuation of hole in dotted lines

Correcting a crooked churn-drill hole in fissured ground

Churn Drilling Fissured Ground

In drilling badly fissured ground with churn drills there is always trouble caused by the swerving of the string of tools out of its vertical course. In the Southeast Missouri lead district this difficulty is overcome by filling the fissured portion of the hole with small rocks and stones of the same kind as the formation penetrated, as indicated in the accompanying figure. This portion of the hole is then redrilled. Occasionally it is necessary to repeat the procedure two or three times, but often the first trial is successful and the abandonment of the hole is thus avoided.



Method of Shafting Alignment

By W. F. Schaphorst

An excellent and inexpensive home-made gage for aligning shafting is shown in the accompanying figure. The gage consists of two pieces of light, slender, stiff, wooden pieces and two clamps. With a gage like this, clamped to the correct length, it is a simple procedure to check up two shafts and determine whether or not they are parallel. If in need of alignment, the gage will touch only at the points of minimum distance. Thus, if the shafts are not parallel the distance between them will not be the same along the entire length, but if the shafts are parallel, the gage will "just touch" at both ends and at every place of measurement.

A wooden gage of this kind is obviously superior to a cord, because it does not stretch or shorten. It is handled with greater ease and certainty and is even better than a steel tape.

Where much shafting is used it is worth while to make a gage of this kind to keep on hand for use as a permanent tool. Fitted with a tongue-and-groove joint the two pieces are adjusted more quickly than without such an arrangement. However, two plain sticks without the grooved joint will serve the purpose very well in an emergency.

Junction of Inclined With Vertical Shaft

By J. P. Crowder
Chief Mine Engineer, The Mexican Corporation, S. A., Fresnillo, Zacatecas, Mexico

The San Francisco shaft, one of the old shafts of the Zacatecas district, is vertical down to the 300-meter level, whence it inclines at an angle of 63 deg. from the horizontal, following the dip of the Agripo vein and also its strike, which is at a horizontal angle of 9½ deg. from the vertical section. This shaft was used by the old operators for unwatering the lower levels of the mine. The pump rod of the Cornish pump was found in position when the 300-meter level was reached. A knuckle joint of 16 by 22-in. timber had been placed at the junction of the inclined and vertical shafts, supported by a cantilever beam, thus enabling the pump to be operated in both inclined and vertical sections of the shaft, from the same engine, which was at the surface.

Although some development work is done from this shaft, its main object is to unwater the mine, as it has access to the deepest levels. The shaft throughout its vertical section, though not of a regular cross-section, averages about 12 by 16 ft. In places it has been walled in with masonry. The present collar is located at the 105-meter level, as the upper part has been destroyed by robbing of pillars by former operators. The hoist is located some distance back in a room cut out of the ground for the purpose. Sheaves and dumping device are placed at proper height to allow dumping into a bin for drawing off at the 105-meter level, whence the material is hauled to the General shaft for hoisting to the surface.

Vertical Portion of Shaft Timbered with Steel I-Beams

The vertical portion of the shaft is timbered with 6-in. I-beam wall-plates, the ends of which extend into hitches in the walls and are cemented in place; the dividers consist of 4-in. I-beams laid on top of the wall-plates and secured to them by bolts. No vertical members are used except at shaft stations. Steel timbering was placed in the center, thus enabling the pumps to operate, one on each side, while sinking in the vertical

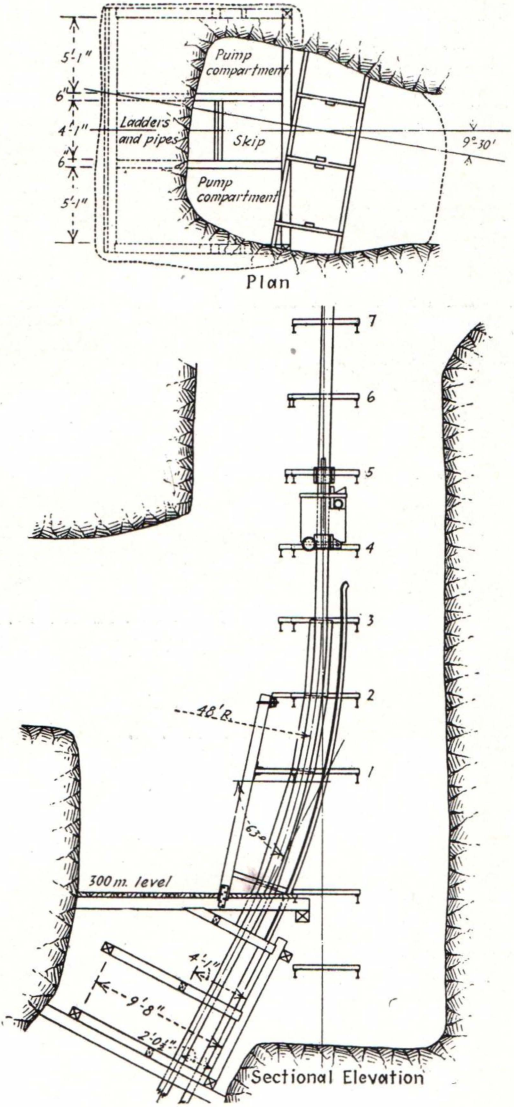


Fig. 1—Vertical cross-section and plan of junction of incline and vertical shaft sections

section. The lower, inclined section is timbered with 8 by 8-in. timber wall-plates and end-plates and with 6 by 8-in. dividers; below the junction, the hanging wall-plate is omitted, as the rock is solid and no lagging is required. The center compartment is lined on the foot-wall with 2 by 12-in. boards. As is shown in Fig. 1, the vertical part is really a two-compartment shaft, while the incline has three compartments. The cage goes only as far as the 300-meter level, where the booster pumps are situated.

As the inclined shaft was under water and full of débris and muck, and there were no accurate data as to its dip or strike, the engineers were extremely fortunate in selecting the proper center line for the incline, for it was necessary to cut out about 2 ft. on either end to allow room for the pump compartments. For the entire length down to the present elevation, the 340-meter level, this center line has been maintained with no change. Had the engineers not been so successful in choosing the center line, it might have been necessary to make an offset to one side or the other, thus destroying the perfect alignment. The Cornish pump rod which was encountered at the 300-meter level gave an idea of direction and inclination; also the center line of the shaft.

The manner in which the horizontal angle of 9½ deg. was allowed for is shown in Fig. 2. It was decided that about 2 in. per set was the maximum amount to turn the shaft (the sets are 6 ft. center to center); and as the cageway was to be omitted below the 300-meter

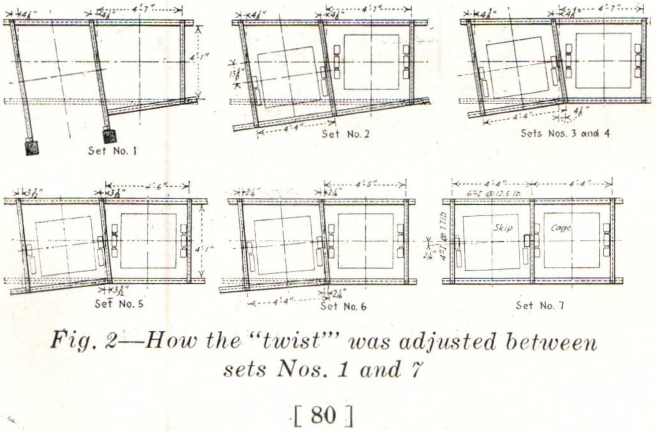

Set No. 1
Set No. 2
Sets Nos. 3 and 4
Set No. 5
Set No. 6
Set No. 7

Fig. 2—How the "twist" was adjusted between sets Nos. 1 and 7

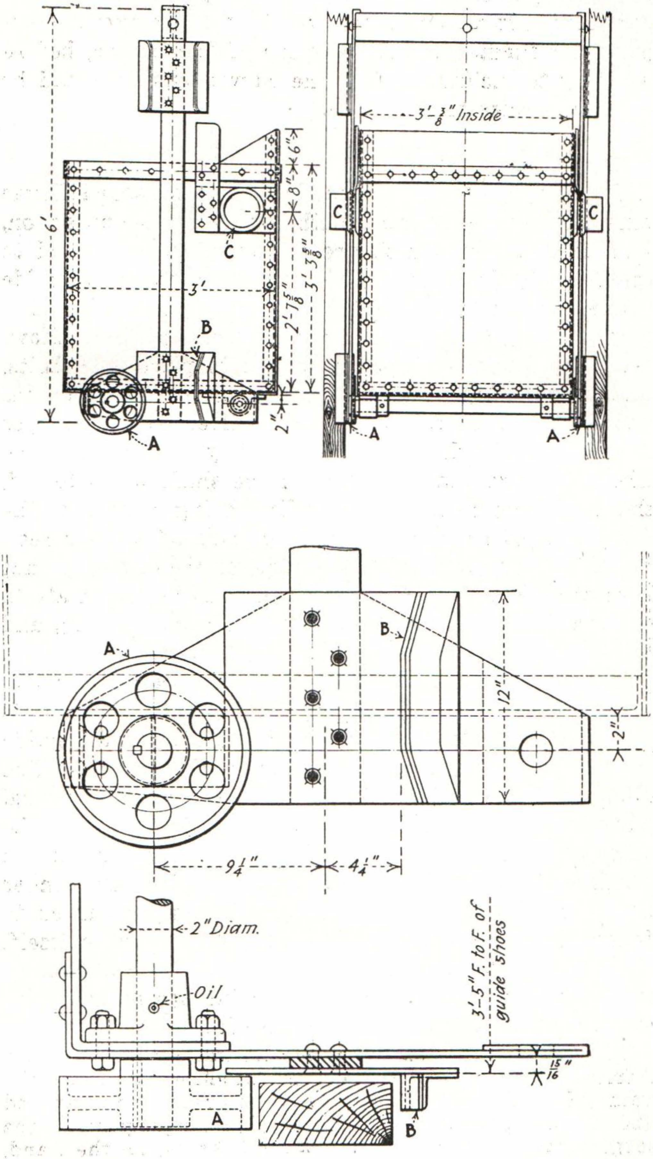


Fig. 3—Details of skip changes

level it was only necessary to change the skip compartment. It was also thought best to bring the compartment normal to the direction of the incline, before starting to incline, so that the curved guides would be at right angles to its strike.

CURVED GUIDES USED

The curved guides are made of well-seasoned Texas pine and are reinforced with, or rather mounted on, ¼-in. steel plates, which are in turn securely bolted to the sets. Difficulty was anticipated in bending the guide nearest the cage, as it is merely twisted, the total distortion being 9½ deg., or 2 deg. 22½ min. per set. However, the guide was readily drawn to place with bolts, and no splitting or splintering resulted. Cage guides and end-plate of cageway were shifted over 1 in. per set, to clear the divider and the safety dogs of the cage. The cuts show how the turn in the shaft was effected, the wall-plate nearest the incline being bent out, the total turn of 9½ deg. being taken care of in four sets. Sets No. 1 and 2 are on the middle of the curve, so that here the dividers were lengthened and the wall-plate in the case of No. 2 was moved out without bending, and on set No. 1 it was omitted in the skipway.

The skip had been used in the vertical shaft and was altered as shown in Fig. 3 by adding a pair of wheels and axle *A*, to take the place of one side of the guide shoe *B* and to act as rollers down the incline. The dumping rollers, *C*, in addition to serving their original purpose, serve as a support for the upper part of the skip when in a slanting position. These latter ride on another set of smaller guides made of 4 by 6-in. timber below the junction and of 3 by 3-in. angles curved to follow the curved wooden guides at the junction itself.

The shaft has been in operation below the 300-meter level for nearly a year, and no trouble has been experienced with it or with the skip since its initial operation.

The premier gold district is the Witwatersrand, in South Africa. In forty years it has produced gold to the value of about £850,000,000. Current production is around 850,000 oz. per month. The mining company with the largest current output is Government Areas, on the Rand, which reported a gross revenue of £386,150 in October. This was equivalent to approximately 90,000 oz. Second place goes to Crown Mines, also on the Rand, and third to the Hollinger Consolidated Gold Mines, of Porcupine, Ontario.

Dump Doors in Winze Sinking

BY C. S. WATT

Assistant Mechanical Engineer, Dome Mines, Ltd.,
South Porcupine, Ontario

During August, 1925, a winze-sinking record was made at the property of the Dome Mines, Ltd. The winze was sunk from Level 13 at a distance 1,608 ft. east of the main shaft. It was 185 ft. deep on Aug. 1 and was sunk to 466 ft. by the end of the month, an advance of 281 ft. The winze was 7x13 ft. and the total tons of rock handled during the month was 2,150.

As in other shaft-sinking records the saving of time becomes an important factor, and if dumping at the surface is slow, progress in sinking is correspondingly diminished. Part of the usual delays at the dumping point are directly attributable to the design and the mechanical working of the dump doors or aprons which are installed to protect the hoisting compartment when the bucket is dumped. In this particular instance of winze sinking a design was worked out, and dump doors, of a type shown in Fig. 1, were installed in the hoisting compartment of the winze. At *A*, the doors are shown in closed position ready for dumping the bucket and at *B* the doors are shown in the open position when the bucket is being hoisted or lowered. As will be noted the aprons or doors are supported by horizontal bars with trunnion supports at the posts on either side of the compartment. The lower apron is constructed with side plates and angles and plates at the top edge of the sides which, in the closed position, rest on top of wooden pieces, thus affording a complete closure when the door is in use. The bucket is dumped into the lower door, the top door serving as a stop for the lower door and also to prevent the bucket being revolved in the wrong direction. As an additional precaution, the swinging lever at the top automatically drops into a locking position when the doors are closed. A counterbalance weight is arranged to reduce the effort required in opening the doors. This comes into play during part of the travel of the doors, as indicated in Fig. 1. The installation worked satisfactorily and proved to be safe and foolproof. One man is able to close the door, dump

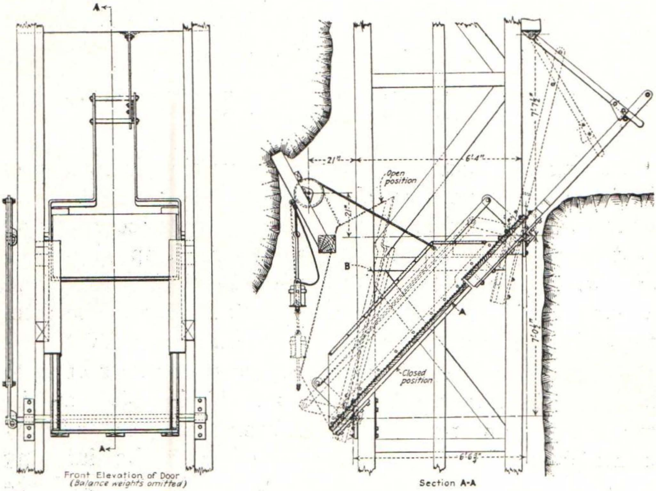
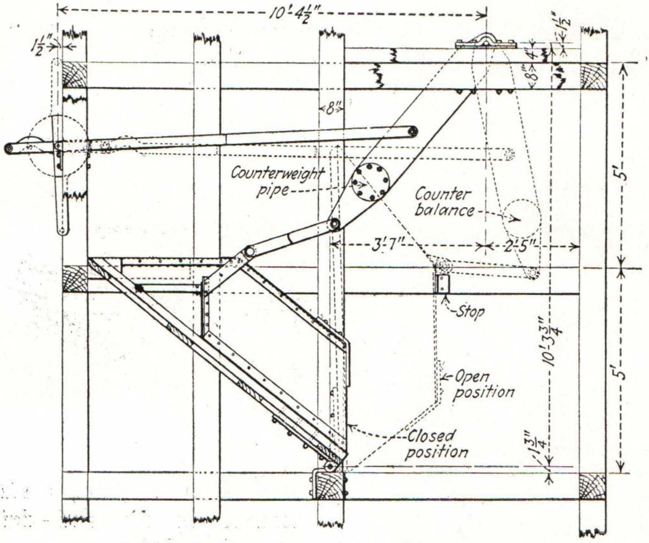


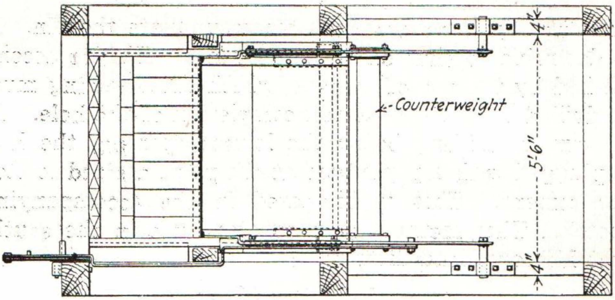
Fig. 1—Double-apron dump door used in winze or shaft sinking. Front and side elevation

the sinking bucket, open and signal to lower in from twenty-five to thirty seconds.

Another design involving a single apron only is shown in Fig. 2. The apron is shown in both the closed position and in the open position. This door was used for sinking work at the No. 3 main shaft and proved to be a safe, quickly operated door. It is provided with a dump plate extension for dumping a larger-sized bucket. The apron is supported upon hinges bolted to the wall plate. It is provided with side plates and the side angles and plates rest upon sealing pieces. In addition it is provided with a removable sole plate. A swinging counterbalance weight is secured inside a steel pipe connected between the two swing arms that support the weight. This form of counterbalance serves the purpose excellently when sufficient clearance for its operation is available, as the effect of the weight decreases and increases in proportion to the weight of the door when opening and closing. Care is necessary, in designing this type, to avoid getting the weight-supporting arms and short connecting links to the door in a straight line, as a strong reverse toggle action takes place and difficulty is experienced in closing the apron. When the apron is closed, it will be noticed that



Part Section Through C.L. of Dump Door



Plan of Dump Door

Fig. 2—Single-apron dump door used in winze or shaft sinking

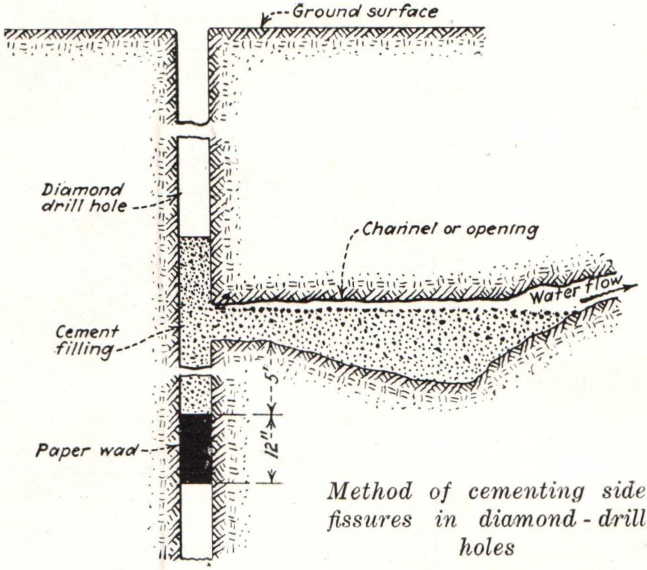
the operating lever arm assumes a dead-center position and remains locked against a stop pin. When the apron is in an open position it rests against two angle stops which also act as buffers.

Cementing Side Openings in Drill Holes

BY ROY H. POSTON

St. Francois, Missouri

Natural caves or crevices that are penetrated by diamond-drill holes allow the mud to flow into the hole,



Method of cementing side fissures in diamond-drill holes

and cuttings from the drill bit accumulate therein. It is desirable to shut off these openings. This is accomplished by the use of cement, which after setting must be drilled through to permit completion of the hole. To prevent re-drilling below the lowest opening, the hole is plugged with a paper wad at the point desired to stop the cement. This is indicated in the accompanying sketch. This method is in constant use in the southeast Missouri lead district.

Recent Powder Magazine Construction

By C. E. Mills
Chief Engineer, United Verde Copper Co., Jerome, Ariz.

Two magazines for storage of explosives have been recently constructed by the United Verde Copper Co. at Jerome, Ariz. One of these is situated 3,000 ft. from the portal of the Hopewell tunnel and is used to store explosives for the mine. It is of wooden construction, covered with corrugated iron, the walls being filled with a 1:8 mixture of cement and sand. The plan, elevation, and section are shown in Fig. 1.

As originally designed, frozen powder was thawed in a small 8x8-ft. room heated by electric grids placed in

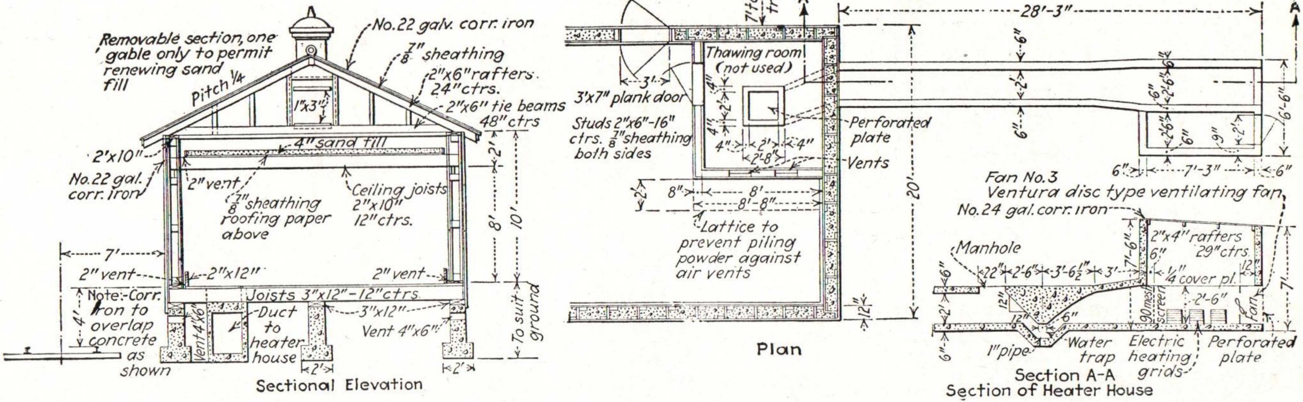


Fig. 1—Plan, section and elevations of United Verde powder magazine

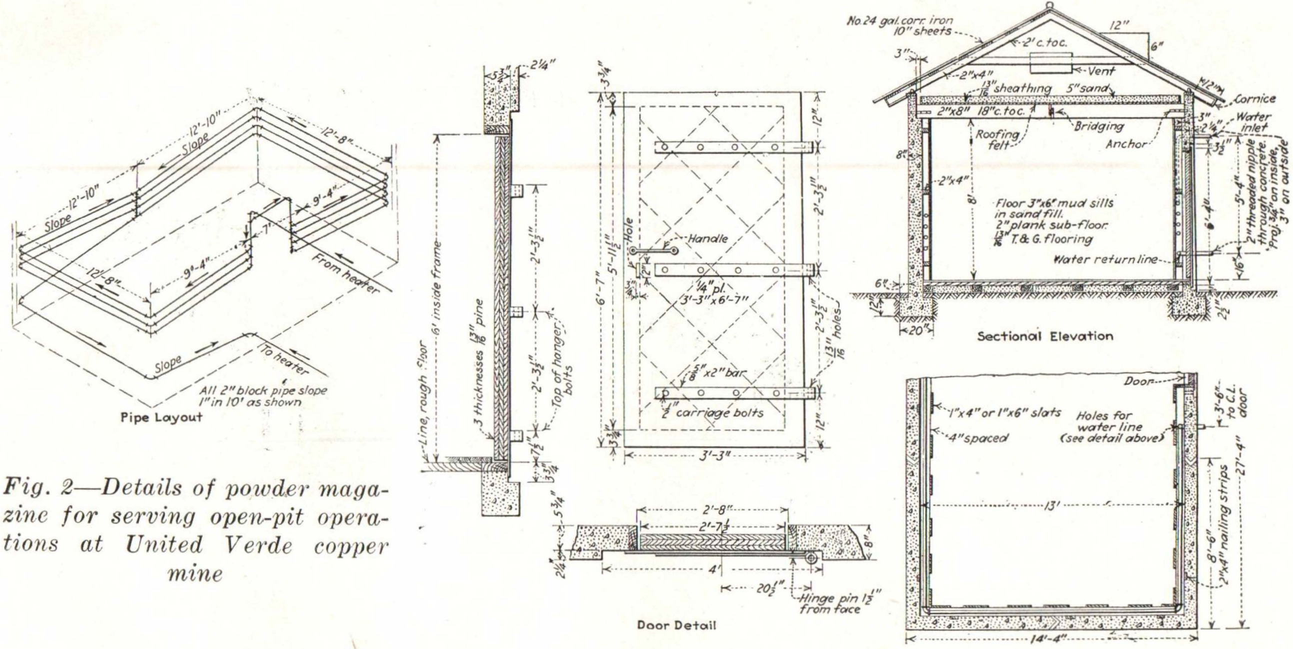


Fig. 2—Details of powder magazine for serving open-pit operations at United Verde copper mine

a small concrete housing 15 ft. distant from the magazine, and connected to the thawing room by a concrete duct. Natural ventilation was depended upon for heat distribution. The old heating unit was below track grade and the encroachment of a waste dump necessitated its removal. The new layout consists of a set of three electric grids inclosed in a concrete housing 20 ft. distant from the magazine. Air is forced over the heated grids by a No. 3 Ventura disk-type fan and passes through a 100-mesh screen and over a bath of water. The screen prevents sparks or hot dust particles from entering the magazine and the water eliminates any possibility of static electricity accumulating in the dry air. Air enters the old thawing room and is distributed to the main portion of the magazine through vents in the wall. A slat partition prevents powder from being piled against these vents.

The second magazine replaced the two magazines destroyed on Dec. 20, 1925, at Hull Springs, on the abandoned U. V. & P. R.R. grade about three-quarters of a mile from the open pit. They were 9x20 ft. inside measurements, with a circular arch roof. The walls were 18 in. thick at floor line, 12 in. at spring line, and 6 in. at center of arch; the spring line was 6 ft. above floor line, giving a 10-ft. 8-in. inside height; the end walls were 12 in. thick; the floor was a 4-in. concrete slab laid directly on the ground. The door was constructed of ½-in. plate with wooden door frame. No reinforcing steel was used and all concrete was a 1 to 7 mixture of cement and sand. The magazines were well designed and the explosion proved the desirability of this type of construction, as no large pieces were thrown when the 950 boxes exploded in the upper magazine.

The new magazine, represented in Fig. 2, replaced the two former ones and the type of construction was changed, as this magazine will be used for only two years and it was inadvisable to go to the expense of concrete construction. The magazine is 13 by 26 ft. inside measurements, with 8-ft. walls of 1:8 cement-sand mortar and a roof of frame construction, covered with 24-gage corrugated iron and a 5-in. sand pad over the ceiling to give protection from stray bullets. The floor is $\frac{13}{16}$ in. tongue-and-groove flooring over a 2-in. plank sub-floor laid on 3 by 6-in. sills.

Special attention was given to safety in heating the magazine, and for this purpose a hot-water system was selected. All inside piping is hung from the walls and entirely covered with vertical wooden slats to prevent piling powder against heating coils. The heating tank is equipped with three 3,000-kw. Everhot immersion type electric heating units, each unit being on a separate switch to provide heat control for varying outside temperatures. The heating plant is housed in a well-insulated shed 20 ft. distant from the magazine, thus preventing any sparks reaching the storage room.

Automatic thermostat control with inclosed contacts was considered in connection with both magazines, but discarded in favor of hand control. Each magazine is provided with a recording thermometer. The powder foreman is responsible for the temperature maintained within the magazine and he endeavors to keep the range between 60 and 70 deg. F.

Drilling and Blasting in United Verde's Open-Pit-Mine

By C. E. Mills

Chief Engineer, United Verde Copper Co., Jerome, Ariz.

OPEN-PIT MINING as practiced by the United Verde Copper Co., at Jerome, Ariz., is a combination of shovel and mill-hole methods for the recovery of orebodies lying between the surface and the 500 level. Mill-hole work has been carried on for possibly fifteen years in the oxide and schist zone at the south end of the present pit. The greater portion of the ore, however, was in the fire zone, and steam-shovel operations were started early in 1919 for the purpose of uncovering this fire area. Pit limits on the 160 level are approximately 1,000 ft. in a north-south direction and 500 ft. in an east-west direction. The ore occurs in the form of massive sulphide, schist, oxide, quartz, and porphyry. The greater portion of the stripping is in a hard, blocky diorite, locally known as United Verde diorite.

Table I shows roughly the yardage removed to date and that remaining to complete the major shovel operations.

The shovel pit as originally planned had eight major

Table I—Yardage Removed to Date in United Verde's Open-Pit Mine

	Yardage Removed	Yardage Remaining	Total Yardage
Steam Shovel Yardage			
Ore	808,000	96,000	904,000
Waste	5,780,000	729,000	6,509,000
Total	6,588,000	825,000	7,413,000
Mill-Hole Yardage			
Ore	83,000	2,069,000	2,152,000
Waste	54,000	1,022,000	1,076,000
Total	137,000	3,091,000	3,228,000
Grand total	6,725,000	3,916,000	10,641,000

levels. The bottom level had a 110-ft. bank, and the remaining banks averaged 50 ft. in height. The greater portion of the stripping was on the west side of the pit in hard diorite. Stripping operations above the 160 level are served by a series of switchbacks on a 3 per cent grade. Shovel equipment consists of one Marion 300-ton, 8-cu.yd. steam shovel and two Osgood Model 120 railroad-type 4-cu.yd. steam shovels mounted on caterpillars. The muck is removed to the waste dumps by 25-cu.yd. Western Wheeled Scraper dump cars.

Below the 160 level, a combination of mill-hole and shovel methods will be used in operation. Equipment for this work consists of two Bucyrus Model 50-B 1¾-cu.yd. electric shovels. Both waste and ore are handled by motor trucks to a raise leading to ore bins on the 1,000-ft. main haulage level of the mine.

In preparing the ground for shovel loading, the drilling and blasting may be divided into primary and secondary operations. Primary blasting is accomplished by churn-drill holes, vertical Leyner holes, jackhammer toe holes, and coyote methods. Secondary blasting consists of plugging and "dobeying" boulders and slabs to a size sufficiently small to be handled by the shovels. Wide variations in the character of the rock made it impossible to formulate any set rules for drilling and blasting which will apply to all cases. The general methods used will be described in more detail.

USE OF CHURN DRILLS LIMITED TODAY

When shovel operations were started, churn-drill blasting was used in removing the entire 110-ft. initial cut and in establishing the various benches or levels. The main advantage of the churn drill over machine drilling is that the depth of hole is not limited as in the case of the Leyner holes. This permits higher benches to be shot and makes possible wider cuts.

No. 5 Keystone well drills, using 6- and 8-in. bits and 16-ft. stems, are used. The rate of drilling varies widely, depending both on character of ground and on the proficiency of the drill crew. Drilling speeds for the various classifications of ground are as follows: Oxide and oxidized quartz, 20 to 30 ft. per shift; schist and soft diorite, 10 to 15 ft.; fresh diorite and jasper, 3 to 5 ft. per shift. The average footage drilled per man-shaft is 2.5 ft. in moderate ground and 0.75 ft. in hard diorite. This includes delays and time taken in moving. The drilling crew consists of a runner, one fireman and a coal boy.

Because the major portion of the stripping is now in hard diorite, the use of the churn drills has become limited. They are now used chiefly for sloping high banks, drilling and shooting hot ground, and in altered diorite or oxidized quartz where the cost of drilling is not excessive.

SPACING OF CHURN-DRILL HOLES DEPENDS ON GROUND AND HEIGHT OF BANK

The location, spacing, and depth of churn drill holes depends entirely on the height of bank and character of material. For medium ground, the allowable burden should never be less than 16 ft. for 20-ft. banks and 50 ft. for 100-ft. banks. Table II gives common practice at the United Verde:

Table II—Spacing and Depth of Churn-Drill Holes

Height of Bank, Feet	Burden on Hole, Feet	Spacing of Holes, Feet
50	28 to 36	32 to 44
110	36 to 44	55 to 95
150	40 to 50	100 to 135

The sub-grade or depth of bottom of hole below the floor of cut is usually 10 per cent of the height of bank being drilled.

In hard ground it is usually necessary to spring the hole three times to form a chamber of sufficient size to take the required charge of powder. In medium ground, twice is sufficient. The powder charge will vary from 600 lb. in 50-ft. holes to 8,000 lb. in 150-ft. holes. In certain instances as high as 10,000 lb. of explosives have been used in one hole. The holes are sprung with 50 per cent gelatin and electric detonators. The amount of powder used in springing will run as follows:

First springing— 1 to 1½ per cent of total powder charge
Second springing— 2½ to 3½ per cent of total powder charge
Third springing— 7 to 10 per cent of total powder charge

Loading Churn-Drill Holes

Apache No. 1 Quarry Special bag powder is used in all dry holes, and 35 per cent gelatin stick powder in wet holes. In the latter case, the sticks are cut into 2-in. lengths before dropping in the hole.

The primer is made up by placing one No. 8 electric detonator in one of several sticks of powder, which are tied together. The lead wires are attached to No. 14 insulated wire and lowered to the bottom of the hole. A string of Cordeau is next wrapped around six or eight sticks of powder and also placed in the hole. A portion of the powder charge is next loaded and another primer similar to the first one, already described, is placed near the top of the charge. The remainder of the charge is then placed and the hole tamped with sand or fine dirt. The Cordeau is then capped with a No. 8 detonator, all three lead wires are connected in parallel, and the hole is shot, using 440 volts a.c. current. If more than one hole is shot at a time, all lead wires are connected in parallel and shot in the same manner.

The powder is loaded into the hole by means of an inclined trough, which allows all loaders excepting the man handling the tamping stick to work at a distance from the collar of the hole. When a shattering action is desired, the hole is loaded up into the barrel; otherwise, the charge is confined to the chamber at the bottom. Jackhammer toe holes are occasionally used to relieve the toe of the bank in case the burden is excessive.

Hole Is Cooled With Water When Hot

In some instances, it has been necessary to shoot holes where the rock temperature is as high as 240 deg. F. The hole is cooled by running cold water into it until tests with a thermometer show that the length of time required to heat up will allow sufficient time for loading and firing. The hole is then sprung, cooled again, and loaded with gelatin powder.

The average cost of drilling churn-drill holes is given in Table III.

Table III—Cost of Drilling Churn-Drill Holes

Churn-Item	Cost per Foot	Cost per Shift
Labor	$2.54	$13.68
Fuel	1.15	6.19
Supplies	0.89	4.77
Repairs	1.37	7.50
Total	$5.95	$32.14

MACHINE DRILLING AND BLASTING

The Sullivan DP 33 machine is used for all plugging and toe holes. All jackhammers are equipped with special air connection which allows full air pressure to be blown through hollow steel. The Ingersoll-Rand 248 machine is provided with handles and used for drilling vertical holes up to 27 ft. in depth. Tripods made of 2-in. pipe, and provided with block and tackle, are used to facilitate handling. Quarter-octagon steel, of ⅞-in. diameter, is used for jackhammers, and 1¼-in. hollow-ground steel for all Leyner holes. Length and gage of drill steel used are given in Table IV.

Table IV—Length and Gage of Steels Used in Drilling Holes by Machine

	Starters			Finishers	
Service	Length, Feet	Gage, Inches	Number, Changes	Length, Feet	Gage, Inches
Plugging	2½	1⅜	..	..	..
Jackhammers	2½	2¼	12	20	1½
Leyners	2½	2¾	19	30	1 9/16

All drill steel is sharpened at the main drill sharpening shop on the 500 level and handled to and from the various benches by means of a White truck, which operates on the standard-gage track. Drill repairs are handled in a small shop near the pit.

MACHINES CHEAPER THAN CHURN DRILLS IN DIORITE

Machine drilling has replaced churn-drill holes in the diorite because of the difference in drilling costs. In hard rock, a Leyner will average 3.7 ft. per hour, whereas a churn drill will average only 0.5 ft.

Vertical holes are drilled from 10 to 15 ft. apart, depending on the ground. The depth will vary from 21 to 30 ft. and the burden from about 16 to 20 ft. Drilling speed will average 3.7 ft. per hour, or 12 ft. per man-shift, with a labor cost of $0.42 per foot. Powder charge will average 112 lb. per hole. The percentage of holes lost by caving and because of broken and stuck steel will run as high as 35 per cent.

Toe holes are used in combination with the vertical Leyner holes and will average 18 ft. in depth. They are spaced from 6 to 15 ft. apart, depending on the character of the bank, and are drilled so that the bottom of the holes will be 2 or 3 ft. below grade.

Drilling speed will average 3.5 ft. per hour, or 17.3 ft. per man-shaft, with a labor cost of $0.22 per foot. Powder charge will average 140 lb. per hole. The percentage of holes lost because of loose ground, broken steel, and similar mishaps will average 20 per cent.

After a series of toe and vertical holes have been drilled, they are sprung with two or three sticks of powder in the bottom of the hole, then with six to ten sticks, and again with eighteen to twenty-five sticks. The chamber after the third springing is usually of sufficient size to hold the required amount of powder.

TOE HOLES LOADED QUICKLY BY MACHINE

Four to six sticks of 50 per cent gelatin are then placed in the hole after inserting a No. 8 electric blasting cap in one of the sticks. The bag powder is then loaded by use of an air loading machine, shown in Fig. 1. The powder is delivered from the gun to the bottom of the hole through a brass tube which permits the air to escape without carrying along a portion of the charge. This loading device will handle 150 lb. of powder in three minutes, whereas formerly about twenty minutes was required to load a 50-lb. box of stick powder. This method applies only to toe holes, the vertical holes being loaded by use of a funnel. After the holes are all loaded, they are tamped with any fine dirt that is handy, connected in parallel, tested with an ohmmeter galvanometer, and shot with 440 volts a.c. current.

A comparison of the yardage, powder and labor costs of the two methods is shown in Table V.

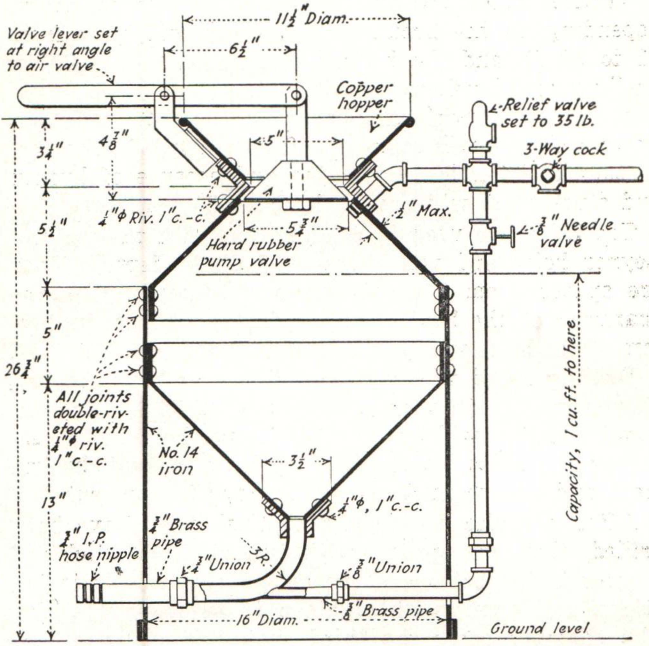


Fig. 1—Powder loading gun for horizontal holes, used by United Verde Copper Co.

Table V—Drilling Costs with Churn-Drills and Machines

	Churn Drilling	Machine Drilling
Powder per cubic yard broken, pounds	0.386	0.398
Cubic yards per pound of powder	3.08	2.99
Cost of explosives per cubic yard broken	90.062	$0.064
Direct labor cost per cubic yard	$0.078	$0.114

Coyote blasting methods are applicable to high banks in waste where no sorting is necessary. Original plans called for stripping the massive diorite from the west side of the pit by a series of 50-ft. benches, and the use of two 4-cu.yd. railroad-type shovels. It was found that stripping operations would delay the opening up of the pit on the 160 level, so that 50-ft. benches were merged into one slope 320 ft. in height on the southwest wall of the pit and broken waste was handled from the 160 level by the 8-cu.yd. Marion shovel, which has

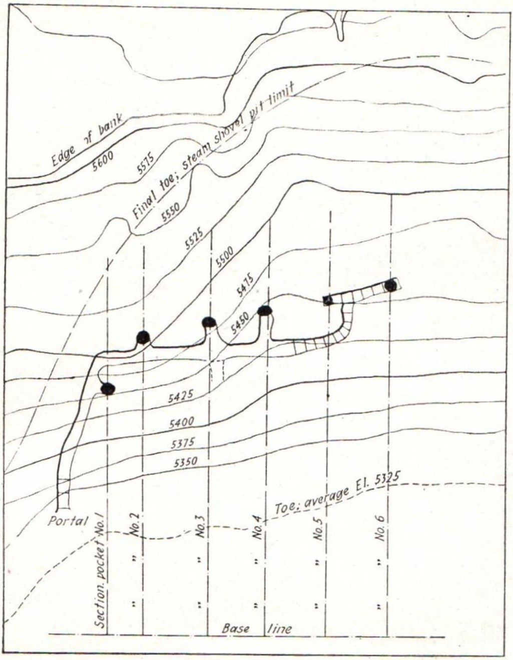


Fig. 2—Plan of development for coyote blast No. 2, at United Verde mine

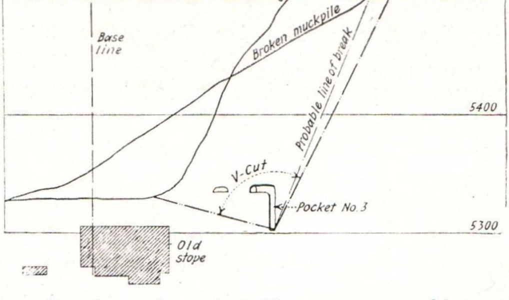


Fig. 3—Section of pocket No. 3, coyote blast No. 2, United Verde mine

a 90-ft. boom and can work against a bank of this height. The general method of development is shown in Fig. 2, and a section through a typical pocket in Fig. 3. To date four such coyote blasts have been successfully fired.

Drifts and crosscuts for coyote blasts are driven 4½ by 6 ft. in cross-section, using Ingersoll-Rand 248 air drills. Pockets for holding the explosive charge are 5x5 ft. in cross-section and are sunk to depths varying from 14 to 30 ft. If a large charge is to be placed in a pocket, the bottom is belled out so that the charge will be concentrated at the bottom of the pocket. Table VI shows the development work done in preparing the four coyote blasts which have been shot to date.

Table VI—Development Done Preliminary to Coyote Blasts

	Number of Coyote Blast			
Item	1	2	3	4
Number of pockets	7	6	14	10
Average distance between pockets, feet	30	30	30	30
Average depth of pockets, feet	15	30	19	16
Drifting and crosscutting footage	305	305	645	526
Sinking footage	105	180	265	160
Average distance pocket to toe of bank, feet	78	100	133	65
Average vertical height above pocket, feet	121	167	189	146
Average area of V-section through each pocket, sq.ft.	4,500	7,760	11,720	4,800
Average burden on each pocket, cu.yd	5,000	7,800	9,800	6,000

It may be noted that in the first three blasts the coyote workings were started from the face of the bank. In the fourth blast all work was carried from behind the bank by utilizing an old adit. Burden, as referred to in the foregoing table, is that yardage above grade, in front of a vertical plane through the rear of the pocket and extending halfway between pockets.

Herco blasting powder was used in the first three blasts. In the fourth Quarry Special, No. 6, manufactured by the Apache Powder Co., was used. Herco powder comes packed in 25-lb. kegs, and the Quarry Special powder is packed in cases, each case containing four 12½-lb. bags. In loading the pockets, the powder was emptied from the containers and trammed to the pocket in special canvas-lines cars. Fifty per cent gelatin was used as a primer in all the blasts in amounts varying from 4 to 13 per cent of the total charge, depending upon the character and structure of the rock to be broken.

Four No. 8 electric detonators are used for capping out each pocket. These are inserted in a box of 50 per cent gelatin and the box is placed in the bottom of the pocket. All lead wires are connected to No. 6 or No. 8 insulated copper wire at the bottom of the pocket. All pockets are connected in parallel circuit. The lead wires are inclosed in old rubber hose and suspended from the back in the drifts. The wires are further protected in the pockets by wooden conduits extending down the sides. As an added precaution against misfire, a line of Cordeau is run through the workings, connecting all the charges and coiled in the bottoms of the pockets. After backfilling is completed the ends of the Cordeau are capped with No. 8 electric detonators and connected to the main circuit.

In loading the pickets the 50 per cent gelatin is first lowered into the pocket and scattered around the primer and the coiled Cordeau. The main charge is then dumped into the pocket. As soon as a pocket is loaded, the explosives are covered with heavy paper and tamped with several cars of sand. Backfilling is then started, the drifts and crosscuts being completely filled with waste material to a point 30 to 50 ft. behind the last pocket. The back-filling is handled in 18-cu.ft. end-dump mine cars. During the progress of back-filling, the blasting circuit is tested every fifteen minutes with an ohmmeter galvanometer, so that any breaks may be readily detected and remedied. When back-filling is completed, the hot is fired instantaneously with 440-volt a.c. current.

Table VII gives comparative data regarding the four coyote blasts.

Table VII—Loading and Back-Filling Time-Powder Used and Yardage Broken in Coyote Blasts

	Number of Coyote Blast			
	1	2	3	4
Loading time, hours	8	6	16	22½
Back-filling time, hours	64	116	136	145½
Cars back-filling	560	995	1,843	956
Pounds black powder	100,000	95,400	250,000	....
Pounds Quarry Special	....	....	....	91,700
Pounds 50 per cent gelatin	13,050	3,500	9,900	11,650
Cubic yards broken	140,000	135,000	259,000	105,000
Cubic yards broken per pound total explosives	1.24	1.36	1.00	1.02

In the first and fourth coyote blasts, the rock in certain areas close to the fire zone was very hot. Great care was taken in the fourth blast not to cut any pockets where the rock temperature exceeded 120 deg. F.

The average cost per cubic foot of rock broken by the four coyote shots is as follows: Cost of development, $0.083; cost of explosives, $0.082; and labor, supplies, and miscellaneous, $0.011; total, $0.176.

SECONDARY BLASTING—BLOCK-HOLING

Secondary blasting consists in breaking up the large boulders into sizes suitable for loading into dump cars or motor trucks. If this can be done without delaying the shovel, the oversize pieces are drilled and shot. This is called block-holing. Where the time element is of more importance than the cost of the powder used, the boulders are "dobeyed"—that is, the powder is laid directly on the piece to be broken and covered with fine dirt and clay or mud. The powder consumption and cost per cubic yard broken for secondary blasting are 0.135 lb. and $0.029 respectively.

Wherever possible, pit operations are placed on bonus, which is handled by one engineer under the supervision of the mine bonus department. Because of the variable ground conditions, it has proved difficult to place the drilling on a footage basis. The bonus rate of all drilling and blasting is now on a yardage basis. These rates are changed from time to time to fit working conditions. At present, the drilling and blasting for the 50-B $1\frac{3}{4}$-cu.yd. shovels are based on a 16c. per cubic yard rate; the 4-cu.yd. shovels, 8c. per cubic yard; and block-holding muck for the 8-cu.yd. Marion is on a 3c. per cubic yard rate.

MILL-HOLE MINING AN OLD METHOD

Mill-hole mining methods have been used at this mine for at least fifteen years. The ore on the southerly border of the fire zone was mill-holed to the 80-ft. level and the schist ore passed down through raises to the ore pockets on the 1,000 level of the mine. Oxide, because of its sticky character, was lowered through No. 4 shaft to the 900 level, where it was dumped into a bin leading to the 1,000 level.

A system of raises and branch raises has been extended from the 1,000-level ore bins to the surface. A number of these will serve the shovels and the rest will be mill-holed where this method is applicable.

In mill-hole work, an initial cut is taken at the bottom of the hole and the cut advanced around the circumference of the hole. Successive cuts are taken, working from the bottom upward until a more or less uniform slice has been taken off of the entire mill hole. This procedure permits holes to be drilled horizontally or upward. It also has the advantage over the method of starting at the top and working down, in that no muck is blasted down on the benches or slope directly below, thereby requiring considerable hand-mucking before the next cut lower down can be drilled.

The drilling and blasting is done in exactly the same manner as described for toe holes. Ordinarily, 35 per cent stick powder is used rather than the bag powder, which requires a gun for loading. The depth of the holes and amount of powder used depend entirely upon physical conditions of the bank being shot and the character of the ground.

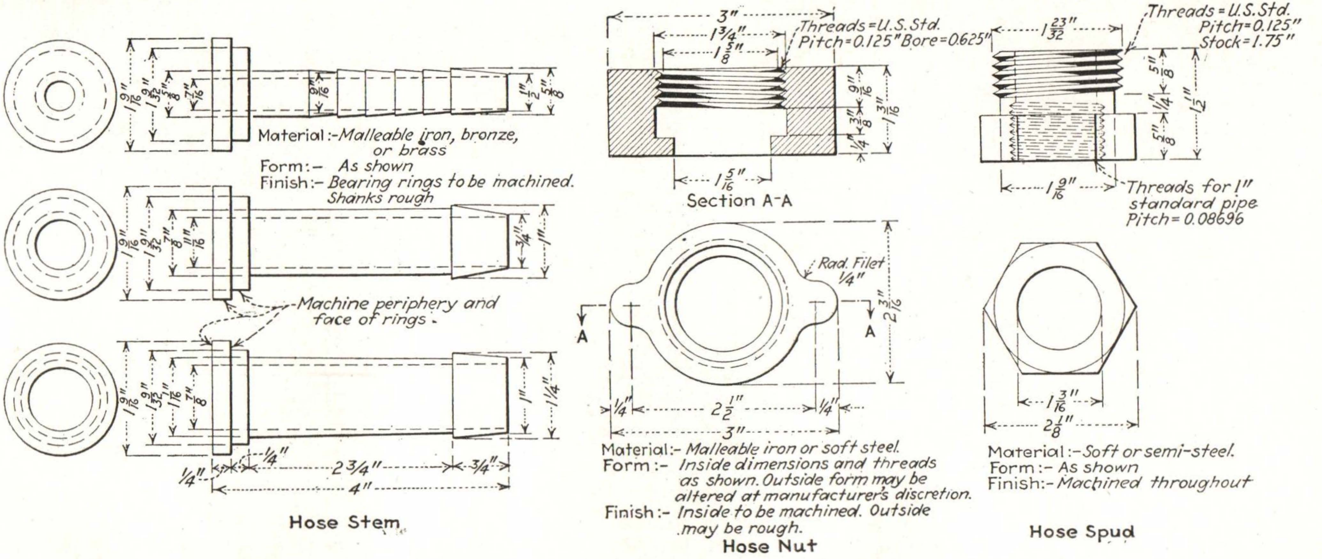
Cost per cubic yard of mill-hole mining is as follows: Explosives, \$0.060; labor, \$0.161; repairs, \$0.039; and miscellaneous, \$0.040; total, \$0.300.

Standard Hose Fittings at the United Verde

By W. P. Goss
Chief Efficiency Engineer, United Verde Copper Co.

Three sizes of hose are in use at the United Verde property—namely, ½-in., ¾-, and 1-in. The 1-in. hose is used for mounted drills, the ¾-in. hose for stopers and jackhammers, and the ½-in. hose carries water to all types of drills. The illustration on next page show how, by the use of three different stems, a standard nut or ring can be used on all sizes of hose to fit a standard spud, this spud being on all types of machines and at the ends of all pipe lines, whether for water or for air. The arrangement allows any size of hose to be used on any drill without changing connections.

Thread on both spud and nut is the standard 1-in. thread adopted by all drill manufacturers for their hose connections. The spud is manufactured by Ingersoll-Rand Co., Denver Rock Drill Manufacturing Co., or Sullivan Machinery Co. The nut or ring has the same dimensions as the standard nut furnished by Denver Rock Drill Manufacturing Co. Stems, nut, and spud can be ordered from any manufacturing concern.



Material:-Malleable iron, bronze, or brass
Form:- As shown
Finish:- Bearing rings to be machined. Shanks rough

Machine periphery and face of rings.

Hose Stem

Section A-A

Material:- Malleable iron or soft steel.
Form:- Inside dimensions and threads as shown. Outside form may be altered at manufacturer's discretion.
Finish:- Inside to be machined. Outside may be rough.

Hose Nut

Material:-Soft or semi-steel.
Form:- As shown
Finish:- Machined throughout

Hose Spud

Details of Standard Hose Fittings at United Verde

II—In Mill, Smelter and Refinery

Recording Alkalinity or Acidity of Flotation Pulp

By D. Welge
Consulting Electrical Engineer, San Francisco

In flotation plants treating complex ores it is important to keep a close check of the alkalinity (or acidity) of the pulp in the flotation cell. Soda is added from time to time in the primary cells in such quantities as to give the desired flotation results. When the pulp contains the correct amount of soda it can be checked by determining the conductivity of the solution, which is indicated by an amount of electric current recorded on a Bristol or similar ammeter. The general arrangement of the apparatus is shown in Fig. 1, (next page).

The plates which are submerged in the flotation cell are of a size and separated from each other so as to give a reading about the center of a 5-amp. scale ammeter chart. This position of the needle will allow a good margin for the swing of the needle according to the variation in the conductivity of the solution in the cell. When too much soda is added the needle will record above 3 amp.; if the current flow as recorded is below 3 amp. more soda should be added. With such chart records the mill foreman can see what has taken place during the night shift or any period when he is absent from the plant. Subordinates have a constant indicator and a record of what has taken place.

It is advisable to install a small 500-w. transformer with separate primary and secondary windings of 110 to 55 volts. Without a transformer the lighting circuit would become grounded through the liquid in the cell and the pipe and other metal equipment in the cell.

The size of the galvanized plates should be about 9x12 in. No. 18 gage. The hanging straps are about $\frac{3}{16}$ by $1\frac{1}{2}$ in. in section; the Bristol ammeter scale 0 to

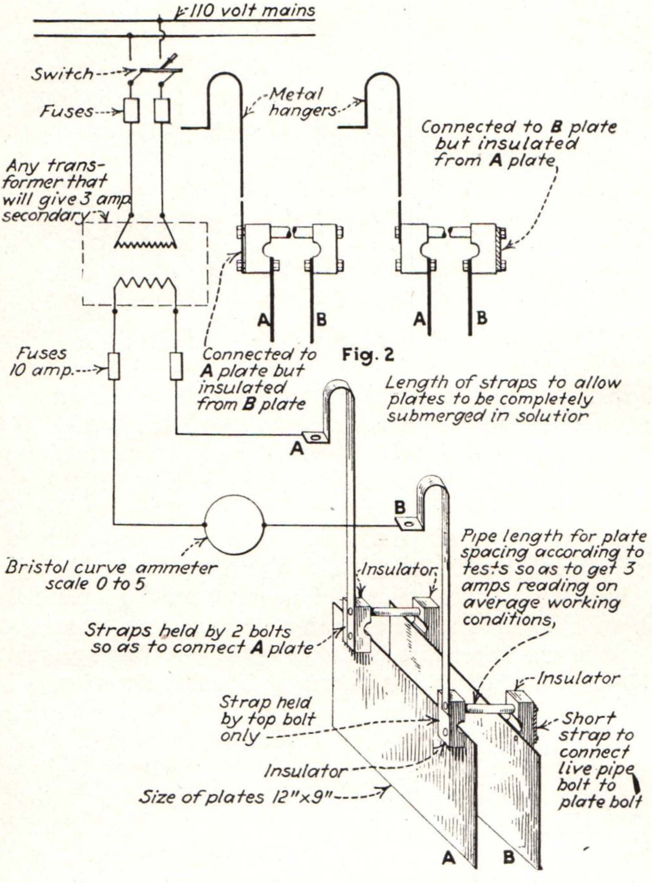

110 volt mains
Switch
Fuses
Metal hangers
Connected to B plate but insulated from A plate
Any transformer that will give 3 amp secondary
A
B
A
B
Fuses 10 amp.
Connected to A plate but insulated from B plate
Fig. 2
Length of straps to allow plates to be completely submerged in solution
A
B
Bristol curve ammeter scale 0 to 5
Insulator
Pipe length for plate spacing according to tests so as to get 3 amps reading on average working conditions,
Straps held by 2 bolts so as to connect A plate
Insulator
Strap held by top bolt only
Short strap to connect live pipe bolt to plate bolt
Insulator
Size of plates 12"x9"
A
B

Fig. 1—General arrangement of electrodes and ammeter used in determining the conductivity of a mill pulp. Fig. 2 (above)—Details of electrode construction

5 amp. twenty-four-hour chart No. 1406. Ten-ampere fuses should be installed in the primary and secondary circuit. The gap space between the plates has been tried out successfully in one plant and found to be about two inches, but as other plants vary in pulp density the space must be ascertained by experiment to give a 3-amp. reading for normal desired conditions.

An Adjustable Drag-Type Classifier

By H. A. Linke
Arthur Plant, Utah Copper Co., Garfield, Utah

In classification operation wherein is used the drag classifier of the Dorr type, the screw classifier of the Akins type, and the Esperanza classifier, a problem is presented which to date does not seem to have been satisfactorily solved.

Owing to variation in the hardness of the ore, it is difficult to maintain a fairly uniform overflow with a minimum percentage of oversize. Where (1) the ore changes from soft to hard and settles readily in the machine, the overflow carries a low percentage of solids and very little oversize; it therefore becomes necessary to reduce the water. Where (2) the classifier feed volume of the classifier were to be reduced, there would be a consequent reduction of undersize material carried over with the sands. In example (2), if the effective settling area and volume of the classifier were to be enlarged, there would result an increase in settling of oversize material, with reduction in the overflow.

Some effort has been expended in the attempt to reduce at will the settling area of drag classifiers by the use of hinged weirs. This contrivance may reduce or increase the setting area about 10 per cent, but the effect on the volume is relatively insignificant. The

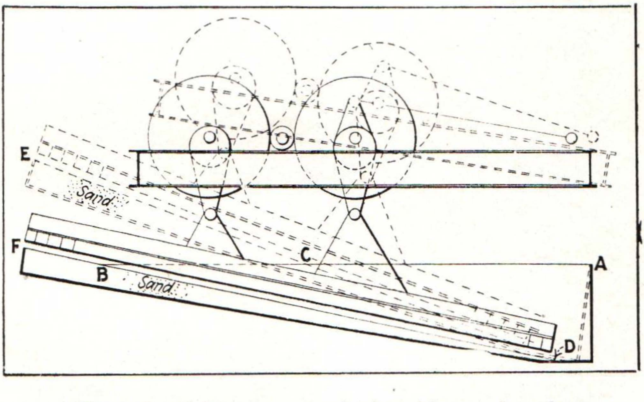


Fig. 1—Adjustment of classifier rakes by tilting raking mechanism

hinged weir does not solve the problem, for the reason that the percentage increase or decrease of settling area or volume is insufficient to produce the desired results.

Increase or decrease in the settling area and volume of the classifier can be simply and quickly effected by tilting the whole machine or its mechanism only, on an axis parallel with the weir lip. Fig. 1 shows in solid lines a drag classifier in the low position, which gives maximum settling area and volume for classification of soft ore. The dotted lines show the whole machine changes also when the tube mill produces an abundance of slime, the feed becomes thick and it is necessary to add more water. The increase in pulp density before addition of water results in a high percentage of oversize remaining suspended in the pulp and being carried

In example (1), if the effective settling area and tilted upward on an axis through the overflow weir at *A*. In this position for operation on hard ore the settling area and volume are reduced to about 60 per cent of that in the first position. Fig. 2 shows the same drag as in the first position of Fig. 1. The dotted lines of Fig. 2 show the mechanism tilted upward on an axis through the weir lip, the pan remaining stationary.

The same results are obtained in both cases. In the second, Fig. 2, with mechanism tilted, the sands form a new bed, and are discharged drier than in the

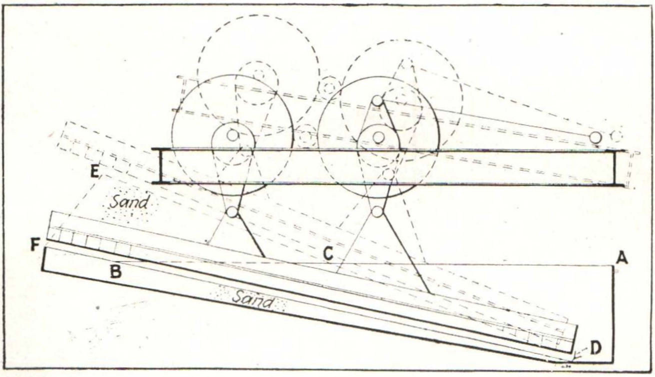


Fig. 2—Adjustment of classifier by tilting classifier about an axis at discharge end

tilted position in Fig. 1. In Fig. 2 less weight is raised than in Fig. 1 and the sand discharge and weir flow remain fixed, thus eliminating practical difficulties in launder construction. It is a matter of choice whether the whole machine or its mechanism alone be adjustable, or where the axis be located. If the axis be through the center of gravity of the machine, or through the center of gravity of the mechanism, as the case may be, the least amount of power will be required to effect the adjustment.

Specific Gravity Indicator in Use at Dome Mines, Ltd.

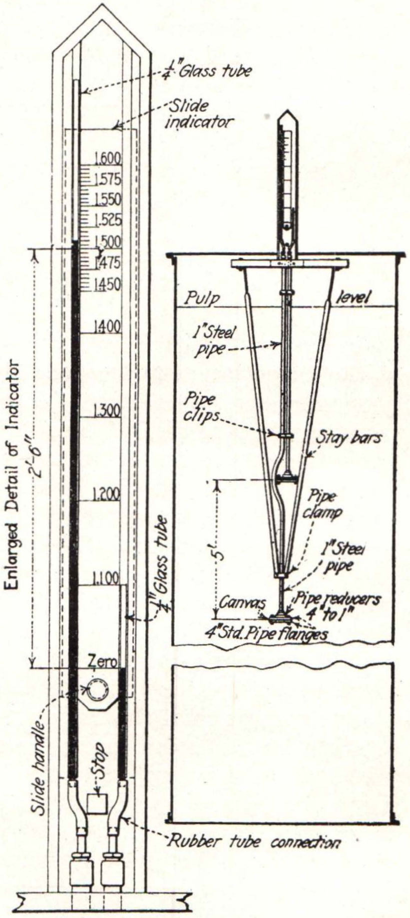
BY C. W. DOWSETT
General Superintendent, Dome Mines, Ltd.,
South Porcupine, Ont.

An hydraulic balance specific gravity indicator in use at the Dome Mines mill consisted of a single tube with necessary attachments, but this arrangement proved to be somewhat unwieldy, inasmuch as it called for a correction of the depth of submergence before a reading could be taken, and this necessitated suspending the apparatus and counterweighting it. A new indicator was designed and installed. It embodies two tubes and is compensating with respect to variation in the pulp level. It occupies a fixed position in the Pachuca tank. In between the two flanges at the lower end of each tube, as shown in the accompanying figure, is a piece of cotton duck which acts as the filtering medium. The submergence of the longer tube can be, say, approximately 10 ft., more or less, while the shorter tube, which is firmly clamped to the longer one, has a submergence of exactly 5 ft. less. When the tubes are submerged in the slime pulp, the clear solution passing through the cotton duck rises up in the steel pipes and into the glass tube to establish the hydraulic balance between the column of slime pulp outside the tubes and the column of solution inside.

The difference in elevation between the solution levels in the glass tubes is the distance that the solution would rise above the pulp level in a tube that had a submergence of exactly 5 ft. A sliding scale is placed at the back of the glass tubes. When the difference in submergence of the pipes is 5 ft., each space of six inches on the scale represents 0.1 in. the specific gravity, so that, if it required to read up to 1.5, the scale must be calibrated in 6-in. spaces for 30 in. Each 6-in. space can be subdivided into tenths to represent 0.01 in. specific gravity.

To take a reading, all that is necessary is to place the lower mark on the scale (representing a specific gravity of one) opposite the solution level in the glass tube of the shorter pipe, when the figure denoting the specific gravity of the pulp is read on the scale at the solution level in the glass tube attached to the longer pipe.

A rise or fall in the level of the pulp in the Pachuca tank affects the solution level in both glass tubes exactly to the same extent, the only factor increasing or decreasing the measurement between the solution levels being the specific gravity of the pulp, as can readily be seen.



Specific gravity indicator for use in cyanide slime vats

Metallurgical Control at the Magna Mill

Metallurgical control at the Magna mill of the Utah Copper Co. comprises weight determination of ore flow, head and tailing samples, water ratio adjustments, screen analyses, moisture determinations, alkalinity determinations, flow adjustments of reagents, inspection of flotation cells, and weight determinations of concentrates. Temperature of pulp is not a factor. Sampling is by automatic samplers, which are controlled by a master timer in such a way as to make sample cuts simultaneously at all mechanical samplers. The cut is made at ten-minute intervals.

No car weights are taken, but occasionally fifty cars of ore car run over the scales and weights are taken. This quantity is run over the weightometers on the main conveyors, and readings are taken, from which a factor is calculated to correct the weightometer readings. These readings are increased about 5 per cent. A grab sample is taken at intervals during the day from the main conveyor belts as a moisture-control sample. This is crushed to ½ in., halved, weighed, and dried sixteen hours at 105 deg. C. Thus a weight estimate corrected for moisture is obtained each day, of the ore delivered to the fine-bins. Mill heads are flotation heads, and samples of the pulp flow are taken automatically for each half of the mill. These are cut down by a splitter. This sample also is the sample used for moisture determination in the mill pulp, which approximates 18 to 20 per cent solids at the rake classifier overflow. A daily assay sample is cut from the head sample and also a portion for a composite monthly sample. Revolution counters on the feeders of each section give a fairly accurate estimate of the tonnage going to the fine-crushing division, and from these and the two daily assays of the heads the mill assay of the heads is calculated for the day. Cleaner tails join the classifier overflow reducing the proportion of solids from 18 per cent to about 15 per cent.

The tailings from each half of the mill are sampled, and likewise a sample is taken from the whole tailings stream as a check on the two large units. These are taken each shift. Partial screen analyses are made, +100, —100 +200 and —200-mesh sizes being determined. Shift samples are only screened to +100 mesh but on daily composites three sizes are determined.

Concentrate samples are taken each shift before the flotation froth is discharged into the thickeners. A composite it is made by combining two portions from each half of the mill. This gives the final daily sample. Portions are cut out for a monthly composite sample. Heads samples are analyzed for iron and copper; concentrates for copper, iron, and insoluble, and tailings for copper. On the monthly composites complete screen analyses are run on heads, tailings, and concentrates and in the analytical work, gold, silver, copper, iron, and insolubles. In addition samples are taken from the filter cake and the thickener discharge, the latter serving to check the percentage of solids and to serve as a measure of thickening-tank conditions. On the filter-cake sample, moisture, iron, copper, and insoluble are determined, which makes possible an estimate on the amount of copper shipped in the concentrates.

The percentage of solids in the classifier overflow varies from 17 to 30 per cent, depending upon the character of the ore. On hard ores the per cent of solids increases and on soft ores greater dilution is necessary. On ball-mill feed solids range from 65 to 75 per cent and are more or less self-regulating. Experience and judgment are the important factors in controlling this feature of the mill operation.

Flow of xanthate solution is fixed, the feeder being adjusted to deliver 0.05 lb. per ton and the quantity ranging from 0.045 to 0.06 lb. per ton. Pine oil is varied to suit conditions and depends on ore. The quantity range is from 0.09 to 0.15 lb. per ton of ore. The appearance of the flotation froth is the test for these reagents.

Control of alkalinity or the maintenance of a fixed excess of alkali is the most important factor. One man on each shift tests the alkalinity each half hour, using phenolphthalein indicator and a standard sulphuric-acid solution. Adjustment of the lime feeder is made as conditions require. The water used in the mill circuit is a neutral carbonate and requires 1½ lb. of lime per ton of ore. An excess of $\frac{1}{10}$ lb. of calcium hydrate per ton of ore, or 3/100 lb. per ton of water, is maintained. This requires from 3½ to 8 lb. of lime per ton of ore.

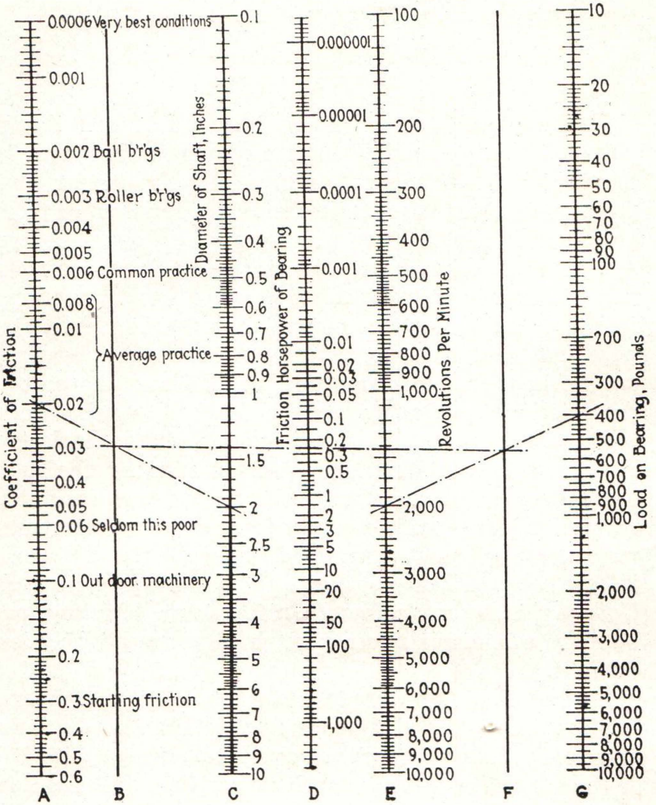
Ball, Roller and Plain Bearings

By W. F. Schaphorst
Newark, New Jersey

If the power is turned on frequently, even though not continuously, and if electric motors are used for driving, ball or roller bearings may be better than plain bearings because the starting friction of plain bearings is high. (See accompanying figure, Column A). On the other hand, the starting friction of roller and ball bearings is not much greater than the running friction. Hence smaller motors may frequently be used with ball and roller bearings. Where plain bearings can be kept in good condition at all times the cost of the frictional loss as compared with ball or roller bearings is often a small item and may as well be neglected. For example, the frictional cost of running the main bearings of an engine, generator, motor, turbine, pump, or similar large unit is ordinarily small as compared with the total power involved, whereas the difference in cost between ball bearings and plain bearings is high. Under such circumstances, therefore, plain bearings may be more economical, even though they are slightly less efficient.

On line shafting where lubrication may be comparatively difficult, the difference in friction may be large and hence the cost of power considerably less with ball or roller bearings—under such circumstances it is advisable to use low-friction type bearings. An important advantage of roller and ball bearings is that they require less lubricant than the plain type and do not require to be oiled so frequently. The total first cost of ball or roller bearings should be compared with the total first cost of plain bearings. Decide on a percentage (30 per cent is considered reasonable by some engineers) to charge against the difference in cost. This percentage should be earned annually by the saving that is effected in the cost of power.

To use the accompanying chart, lay a straight-edge across it three times, as indicated by the dotted lines, and it will give the horsepower required by any bearing; coefficient of friction varying from 0.0006 to 0.6; shaft diameters from 0.1 to 10 in.; r.p.m. from 100 to 10,000, and the load on the bearing from 10 to 10,000 lb. As

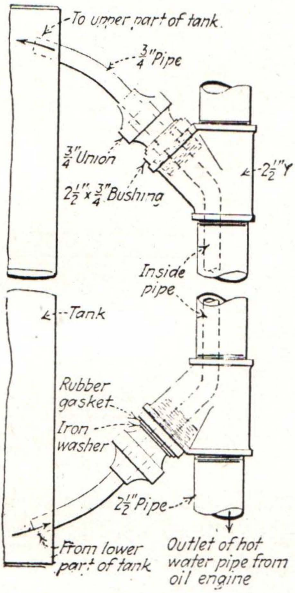


Bearing factors and their magnitude

an example, by using a coefficient of friction of 0.02 (Column A), (average practice for plain bearings), a shaft diameter of 2 in. (Column C), a speed of 2,000 r.p.m. (Column E), and a 400-lb. load on the bearing (Column G), Column D gives the friction horsepower as a little over 0.25. Connect the known value in Column A with that in Column C and locate the intersection with Column B. In the same way connect columns E and G and locate the intersection with Column F. Then connect the intersections (Columns B and F) and the intersection with Column D gives the friction horsepower of the bearing.

Keeping a Solution Tank From Freezing

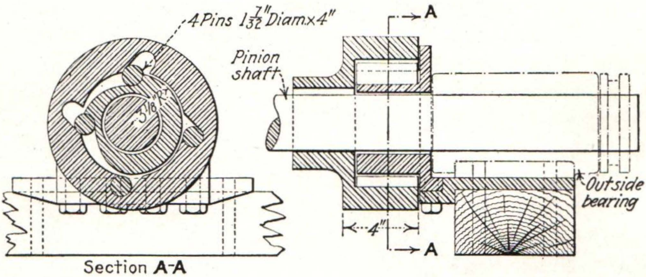
BY CHARLES LABBE



Detail of Sleeve Piping Used to Warm Solutions

Last winter it was necessary to keep a solution tank from freezing. Many experiments were made, but the one here described proved the simplest and best. The plant was operated by an oil engine, the water-cooling outlet pipe of which was altered as follows: The long piece of 2½-in. pipe was taken out and cut in two places to permit placing in the line two Y fittings, as shown in the accompanying figure. A ¾-in. pipe was run from a point near the bottom of the solution tank and another connection of the same size made on the tank about one-third the distance from the top, the pipe being extended to each Y fitting. From one Y to the other a piece of pipe of the same size was placed inside of the hot-water pipe, thus forming a thermo-syphon which kept the solution from getting to the freezing point.

To make the connection at the Y, the inside pipe was cut off to the required length after trying it in place, the threads at each end being cut about ¾ in. Into the small openings of the bushings a tap was extended from the inside to about half way, so as to give some taper to the threads on the reverse side, but still not making the hole larger than necessary. After slightly bending both ends, the ¾-in. pipe was placed inside the large one, the ends extending through the Ys; the bushings were screwed on the small pipe to about four or five threads and then into the Y's when tight, not less than ¾-in. of the ¾-in. pipe extending beyond the bushing. A rubber or leather washer was then put over the ¾-in. pipe and a light iron washer over this. A ¾-in. union was then screwed on the ¾-in. pipe, the half near the bushing acting as a lock nut and preventing leakage. The connections to the tank were made next. It was found necessary to insulate the ¾-in. pipe by strips of old canvas wound around the pipe and held with adhesive tape where needed.



Lock for Inclined Conveyors

A novel type of conveyor lock designed by the engineering staff of the Utah Copper Co. for use upon inclined belt conveyors is shown in the accompanying illustration. The lock is mounted upon the pinion shaft close to the outside bearing. The outer revolving cup is keyed to the pinion shaft and rotates with it as shown in section *A-A*. On the inner surface are four depressions which contain four cylindrical pins. As each pin is carried over the stationary sleeve the pin drops into a curved run which returns the pin to its compartment. A second curved runway is placed in the lower left quadrant of section *A-A*. The pin falls into this and is again returned to its compartment. When the pinion shaft is stopped it is prevented from reversing its direction of rotation by the pins catching against the shoulders of the curved runs on the stationary sleeve. The device was planned for belt-driven conveyor units. but was never actually applied.

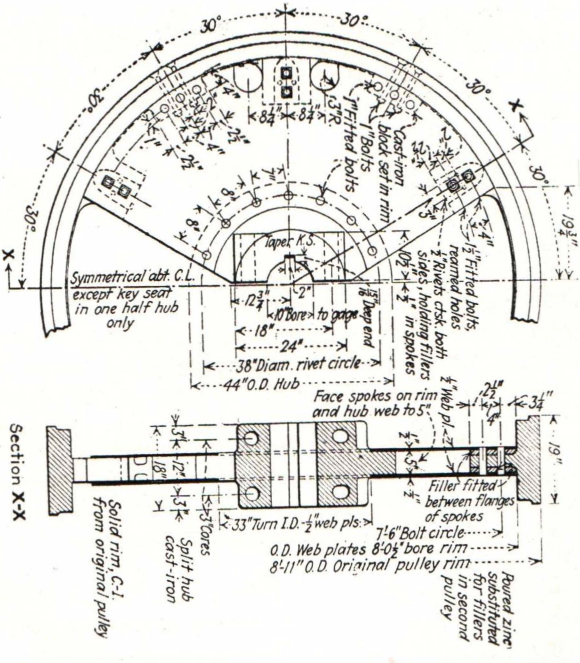
Whether it would stand severe operating conditions or would function satisfactorily is an open question. It is here presented as an illustration of an ingenious suggestion only.

Repair of Roll Fly Wheel Pulley

By W. J. Tait

The roll pulley shown in the accompanying illustration is a web center type rebuilt from the salvage parts of an older pulley. The older pulley was of cast iron with a split hub to facilitate removal from the shaft when changing shells. Its arms were of I-section and had failed by cracking near the hub. It seemed that a web center would overcome such trouble, and this design was made to utilize the old rim and certain plate sizes in stock.

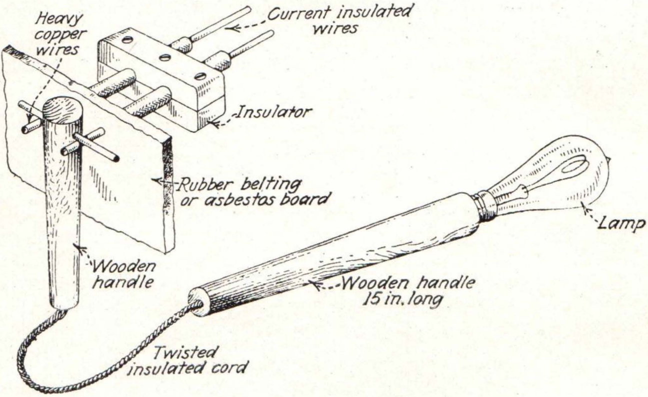
A new hub in two parts was cast with a central flange; the pulley arms were cut off and faced on both sides for double web plates; and fillers fitted into the arms and between them to allow the attachment of the web to the rim. Holes for the attachment of slings for handling were left in the webs as shown. Their location was determined by certain clearances and positions



30°
30°
30°
30°
30°
30°
X
X
4"
2"
1"
2½"
4"
2½"
8¼"
8¼"
3"R.
⅞" Fitted bolts
1" Bolts
Cast-iron block set in rim
7"
8"
8"
3"
4"
1½" Fitted bolts, reamed holes
¾" Rivets ctsk. both sides, holding fillers
½" in Spokes
10½"
19¾"
Taper K.S.
Symmetrical abt. C.L. except key seat in one half hub only
12¾"
2"
10" Bore to gage
15/16" Deep end
18"
24"
38" Diam. rivet circle
44" O.D. Hub
Face spokes on rim and hub web to 5"
¼" Web pl.
2½"
4"
3¼"
19"
Section X-X
3"
3"
18"
12"
½"
5"
½"
Filler fitted between flanges of spokes
3" Cores
33" Turn I.D. ½" web pls.
7'-6" Bolt circle
O.D. Web plates 8'-0½" bore rim
8'-11" O.D. Original pulley rim
Solid rim, C.-I. from original pulley
Split hub cast-iron
Poured zinc substituted for fillers in second pulley

Details of design of built-up flywheel pulley demanded by repair operations. The hub is clamped together by four $2\frac{3}{4}$-in. bolts.

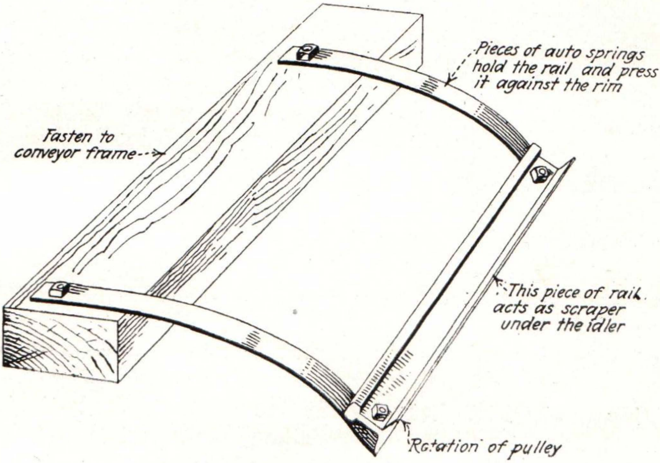
This pulley has been in regular use for the last three years, and another has been made in the last few months. The only change in the later one is the substitution of poured zinc for the fillers between the flanges of the old pulley arms. This seems to give equal satisfaction with less labor and difficulty in fitting up the fillers.



Handy Electric Light for Stamp Batteries

At the stamp mill of the Argonaut Mining Co., at Jackson, Calif., a convenient portable light is used to examine the interior of the mortar box. Between each pair of batteries a lighting circuit is attached to two bare copper wires of heavy gage which are supported at a fixed distance apart by a heavy piece of discarded rubber belting, as shown in the accompanying sketch. The contact points are supported on the under side of the cam floor, which is above the front of the batteries. The bare projecting wire stubs are therefore sufficiently high to avoid any danger of contact.

A short length of twisted insulated cord is attached to a wooden handle from the end of which project two bare wire stubs. The handle is of sufficient length to enable the battery man to hook the points upon the wire stubs above the battery. The other end of the cord is attached to a wooden handle which carries a socket and lamp. The length of cord is just sufficient to enable two neighboring batteries to be examined conveniently. A small panel is placed in the top of the screen frame, and through this opening the light can be introduced and inspection of the stamps and mortar made.


Pieces of auto springs hold the rail and press it against the rim
Fasten to conveyor frame
This piece of rail acts as scraper under the idler
Rotation of pulley

Scraper for Conveyor Pulley

By Charles Labbe
Clarkdale, Ariz.

In wet weather it will be found that much fine material adheres to conveyor belts and also to the larger idler pulleys of the return belt, causing in a short time objectionable accumulations. To get rid of this material scrapers are usually placed at the first large idler. The simplest scraper within my experience is made of a piece of light rail weighing from 8 to 12 lb., of the same length as the face of the pulley and pressing very lightly against the rim by means of two pieces of discarded auto spring. At each end the rail is bolted to the spring, and this in turn is fastened to the conveyor frame by lagscrews or bolts as indicated in the accompanying figure. The springs keep the scraper against the pulley's rim without further adjustment.

Sampling of Concentrates in Bins

BY DONALD F. REED

Ore-Purchasing Agent, The Southwest Metals Co.,
Humboldt, Ariz.

The Southwest Metals Co., operating a custom copper smelter at Humboldt, Ariz., closed down in 1924. During the shutdown period the company leased its concentrator to the United Verde Copper Co., Jerome, Ariz., for the purpose of making an experimental test run, preliminary to construction of a concentrating plant by the last-named company. In the course of this test run about 1,300 tons of flotation concentrates were produced and stored in the Southwest Metals' bins at Humboldt. When the Southwest Metals resumed operations, in 1925, it agreed to purchase these concentrates from the United Verde Co.. It therefore became necessary to determine the tonnage and grade of the concentrates. This was done by sampling with a pipe and auger.

The pipe-and-auger method of sampling finely ground material is not new. The unusual feature in this case lay in the fact that not only the grade of the material was ascertained, but the specific gravity and tonnage of the concentrate as it lay in the bins were also determined directly from the sampling. Sampling was done under the supervision of D. F. Reed and W. C. Goeglin, of the Southwest Metals Co. and was watched and checked by A. L. Reese, O. E. Young, and J. Burkett for the United Verde Co.

The capacity of each bin was first determined by careful measurement. As the bins were full, it was necessary to take the measurements on the outside of the bins, allowing for the determined thickness of wall and floor plates, slope of bottoms, and volume of louvres. Deductions were then made for known voids in the tops of the bins. These were determined by measurement and inspection.

The weight of dry material per cubic foot was then found by sinking holes of known diameter and depth to the bottom of each bin, carefully removing, drying, and weighing all of the material from each hole. As this was weighed before and after drying, the percentage of moisture was thus found.

Then the weight of dry material per cubic foot may be designated as $W = \dfrac{4M}{\pi D^2 \times H}$

Where M = Weight of dry material taken from pipe.
D = Diameter of pipe (feet).
H = Depth of hole (feet).

As a pipe exactly two inches in diameter was used, this formula resolved itself into:

$$W = \frac{45.87\, M}{H}$$

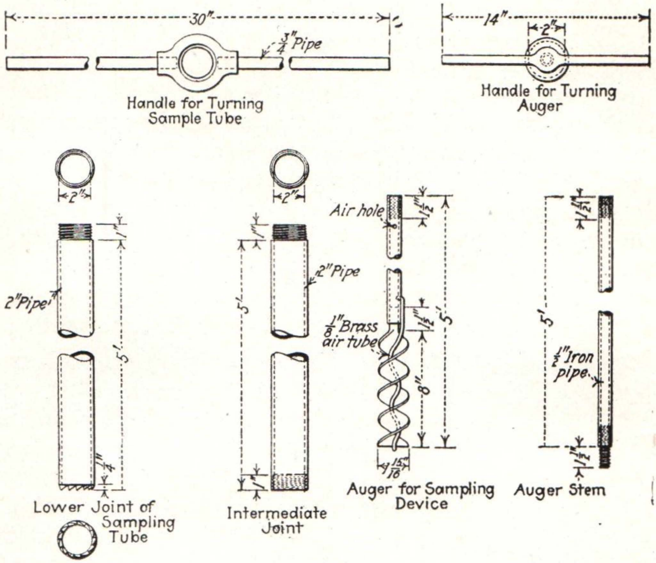
The material taken from the pipe was then made up into pulps for assaying in the usual manner.

The concentrates were in three separate bins and a pile on the distributing floor. They were therefore sampled and considered as four lots, four samples being taken in each of two bins, two samples in the third bin, and two samples in the pile on the floor. In calculating the contents of each lot, an average was taken of all the samples in that lot. With the dry weight per cubic foot and the volume of each lot known, the tonnage was then computed. An average of the assays in each lot was then taken and the metallic content calculated.

The samples did not check very closely as to weight of material per foot, nor was it expected that they would; as the material was in some places noticeably denser and more compact than in others, the moisture content varied, and there were undoubtedly air spaces and voids throughout the mass. The moisture content varied from 6.62 per cent in one lot to 14.31 per cent in another, the other two lots containing 10.94 per cent and 11.50 per cent moisture respectively.

Final results gave a total of 1,337.556 dry tons, containing 32.016 oz. of gold, 2,530.83 oz. of silver, 298,899 lb. of copper, 8.97 per cent "insoluble," and 31.75 per cent iron. The gross value of these metals as determined by this sampling checked within less than one-half of 1 per cent of the value as calculated from the metallurgical reports on the mill operation.

This method could be used to sample any fine-grained material, such as concentrates or tailings, in either bins, ponds or piles, and affords a much closer check upon



Sketch of pipe and auger for sampling fine material

tonnage than usual methods of estimation. The accuracy depends upon the number of samples taken and the care in estimating the volume of the material sampled.

The sampling was done with a wrought iron pipe of uniform inside diameter of two inches. This pipe was in five-foot lengths, to enable the samplers to handle it within the confines of the distributing floor. The ends of each section were threaded so as to connect with other sections. The first section of this pipe was beveled and serrated on the bottom end to facilitate its cutting down through the material. A detachable handle with which to turn the pipe was provided.

The ore or material from the inside of the pipe was removed by means of an ordinary wood-auger attached to a rod of the same length as the pipe, a section of this rod being added each time a section of the pipe was added. This auger was of such a diameter as to fill the pipe entirely, leaving only enough clearance between pipe and auger to permit the removal of the latter. The length of the rod was such that the cutting edge of the auger was always exactly even with the cutting edge of the pipe. A detachable handle for the auger was also provided.

In order to be sure of obtaining all of the material within the pipe, it was found necessary to keep the auger turning faster than the pipe. It was also found that the core had to be removed from the pipe every six to fourteen inches, depending on the compactness and wetness of the concentrates. In the wetter material, the friction on the sides of the pipe, and the vacuum produced when the auger was withdrawn, pulled all of the core off of the auger except that carried in the spiral of the auger itself. To reduce this tendency, a small brass pipe of approximately ⅛ in. inside diameter was run from the tip of the auger up around the spiral and tapped into the center of the hollow rod. Upon pulling the auger up, the suction cleared this pipe and allowed the air to break the vacuum.

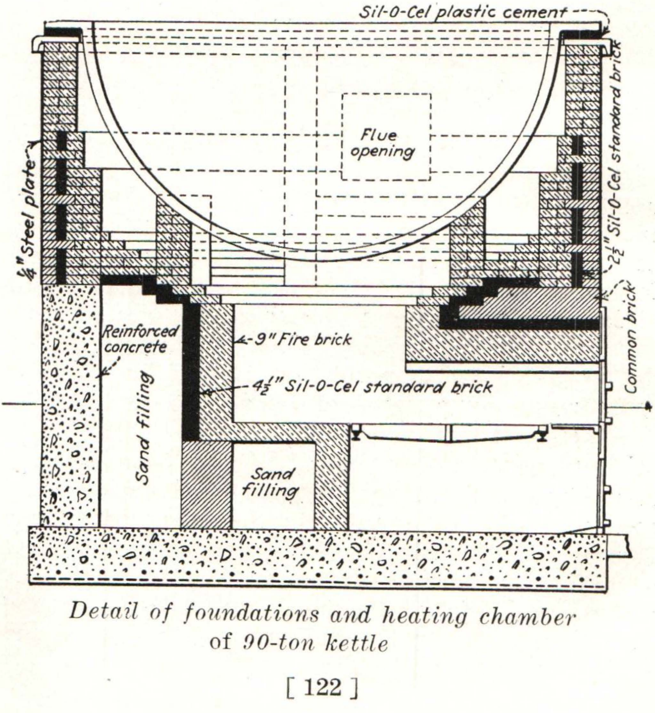
A 90-Ton Lead-Remelting Furnace of Improved Heat Efficiency

By F. M. Wichman
Plant Engineer, U. S. Smelting, Refining, & Mining Co.,
Midvale, Utah

The accompanying sketch shows a cross-section of a 90-ton remelting furnace for lead bullion, and the method of insulating the heating chamber. This furnace was designed and constructed at the Midvale smelter of the United States Smelting, Refining & Mining Co. about a year and a half ago, and in that period of daily use it has not been found necessary to make any repairs to the furnace. When full, the capacity of the kettle is more than 100 tons of lead, but to allow space for the accumulation of dross the kettle is seldom filled nearer than 4 in. from the top.

The construction is somewhat different from the usual 30-ton furnace in common use among lead smelters. On account of the great weight to be supported, it was thought advisable to carry it on steel columns instead of on the brickwork of the furnace. On the foundation, which is of heavily reinforced concrete, were erected six 6-in. x 14.75-lb. I-beam columns equally spaced and properly located so that their flanges were flush with the outside perimeter of the furnace wall. These columns are 11 ft. 5 in. long and support a heavy cast-steel ring, which, in turn, supports the kettles. The total weight supported by the columns is upward of 125 tons. This includes the weight of the kettle, ring, and stirring machine, in addition to that of the molten lead.

The circular wall of the furnace is built of concrete, 16 in. thick, for a distance of 5 ft. 9 in. above the foundation. The columns, which are practically buried in the concrete serve as reinforcement. From the concrete up to the ring the wall is constructed of brick, with common brick on the outside, firebrick on the inside and Sil-O-Cel brick between. The interior of the furnace—that is, the firebox—is constructed of firebrick completely surrounded by Sil-O-Cel brick. The space between the Sil-O-Cel and the concrete wall is filled with sand or granulated slag. A circular steel shell made from ¼-in. plate is riveted to the outside flanges of the columns and completely encases the brick wall of the furnace.



Detail of foundations and heating chamber of 90-ton kettle

The grate area of the furnace is identically the same as that of the 30-ton furnace—namely 2 ft. 0 in. x 4 ft. 7 in. This has been found amply large in spite of the greatly increased melting capacity. The outside of the furnace remains at approximately the same temperature as the surrounding atmosphere, except near the top, where the brick wall is too narrow to permti Sil-O-Cel insulation. It has been found that owing to the high efficiency of the insulation, this furnace consumes no more coal in melting 90 tons of lead than the uninsulated furnaces consume in melting 30 tons. The only difficulty in operating such a large furnace is to cool the bullion to the proper temperature for casting. This is done by sprinkling the surface of the molten lead with water after removal of the dross.

Lime Circuit at Magna Mill

At the Magna mill of the Utah Copper Co. lime is ground in a conical ball mill in closed circuit with a rake classifier, the overflow being piped to two large steel storage tanks in the upper part of the fine-crushing division of the mill. A twenty-four-hour supply is maintained. The tanks are equipped with slow-moving agitators on vertical shafts driven by vertical motors and speed-reducing gears. The two tanks are equipped with a pair of direct-driven centrifugal pumps, which discharge into a common launder over the lime feeder and maintain a constant level of milk of lime in the feeder box. The feeder box contains a horizontal shaft carrying a series of metal disks; on both sides of each disk, except the end disks, are metal buckets which dip into the lime hydrate solution and discharge the lime into separate launder outlets. These end disks are provided with buckets on one side only. The shaft is driven by a motor and speed reducer. The speed reducer permits of twelve variations in the speed of the bucket shaft. Wooden launders convey the lime solution to the bucket elevators receiving the undersize of the impact screens. The mill circuit is tested every half hour for alkalinity and the feeder adjusted accordingly. One man on each shift makes the alkalinity tests and feed adjustments. Tests are made with phenolphthalein indicator and a standard sulphuric-acid solution.

Compressed-Air Shovel

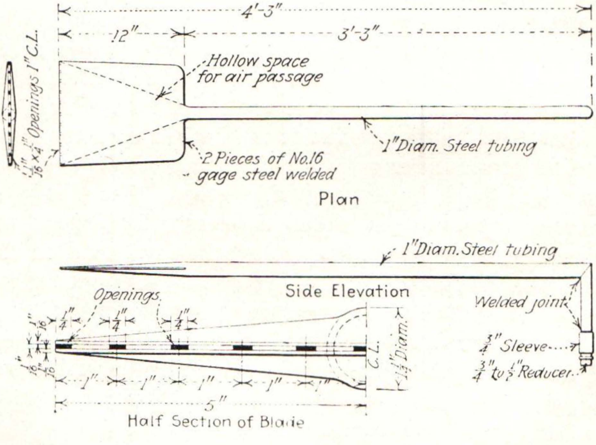
By J. Owen Ambler

Smelter Superintendent, Phelps Dodge Corporation,
Clifton, Ariz.

A compressed-air shovel, devised by T. A. O'Neil, foreman in the preparation department at the Clifton smelter of the Phelps Dodge Corporation, has proved especially suitable for unloading concentrates. The concentrates are received from No. 6 concentrator at Morenci in inverted V-bottom cars equipped with side doors. The construction of the shovel is shown in the accompanying drawing.

By use of these shovels the size of the unloading crew has been cut to half, or less, of that formerly used and the unloading operation much expedited. In a test one man using the compressed-air shovel, with air at about 75-lb. pressure, unloaded and thoroughly cleaned a car containing forty-five wet tons of concentrate in seven minutes, including one spotting of the car.

Under regular routine operating conditions where two men unload a string of from six to ten cars of concentrate, the time for the two men averages ten minutes per car, exclusive of spotting, making the labor cost for unloading, including thorough cleaning of the car, about



Plan, section, and design details of compressed-air shovel one-third of a cent per wet ton. The concentrate handled here is about 60 per cent flotation and 40 per cent sand, and contains 9 to 11 per cent moisture and 22 per cent copper. On account of its high copper content particular attention is paid to thorough cleaning of the cars, to prevent loss in transit when they are returned to the concentrator.

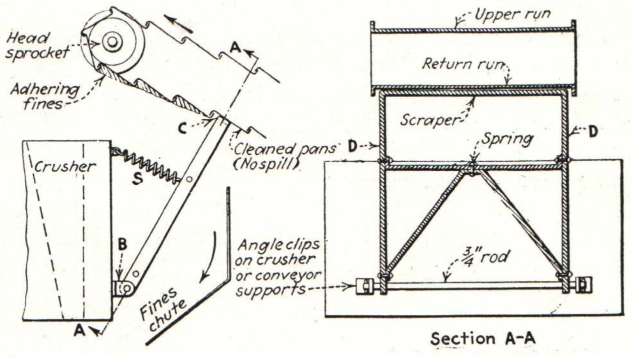
Concentration of Solution Values in Evaporating Cyanide Pulp Samples

By G. C. Tyrrell
Engineer, Kirkland Lake Gold Mining Co., Ltd.

On assaying pulp samples, from the cyanide plant, containing high-grade solution some difficulty was found in obtaining concordant results. With similar samples containing low-grade solution no difficulty was experienced. Several pulp samples containing high-grade solution were taken, quartered down, put in enamel-ware dishes, and evaporated to dryness on an electric hot plate. Before evaporation an assay was made of the solution contained in the sample and the moisture was carefully taken. From these results the gold in the solution part of the sample was calculated. The final dried pulp was assayed and was found to contain less gold than was actually contained in the evaporated solution.

A very thin layer of dried pulp was noticed on the bottom and sides of the dish in which the evaporation took place. This was found to assay more than ten times as much as the original solution. From this one can see that the gold in solution concentrates on the bottom and sides of the dish (where the heat is greatest).

The remedy in use here is to add a small quantity of zinc dust to the pulp before evaporating. In spite of the muddy condition, this is found to reduce a $3 per ton solution to 15c. per ton. The pulp may then be evaporated to dryness with little or no concentration.



Scraper for Beaded Pan Conveyor

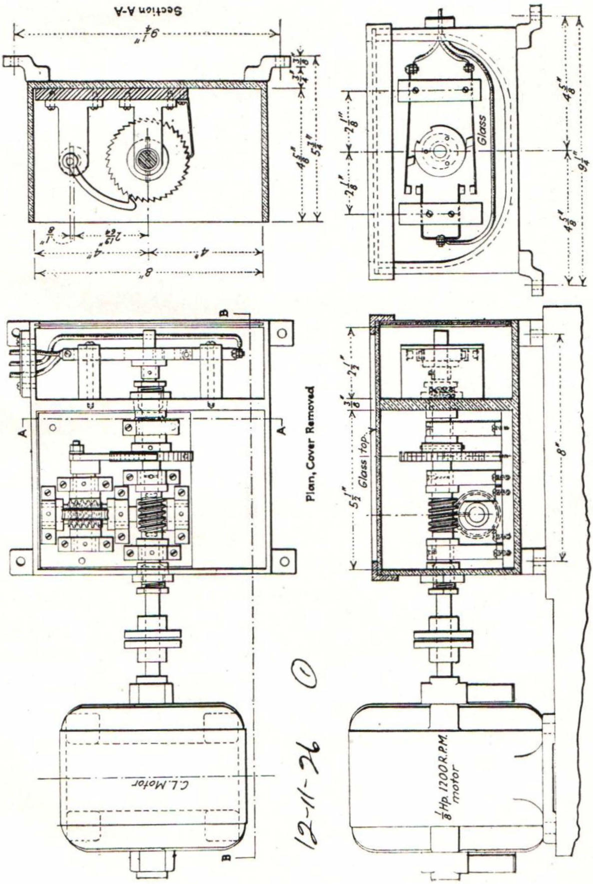
By Jesse C. Stoddard
Dover, N. J.

Where beaded pan conveyors or feeders are used to feed crushers, it frequently happens that when the material is of a wet or sticky nature a considerable amount of the fines will stick to the return run of the feeder and cause a troublesome accumulation of the material beneath the feeder, which requires periodic cleaning up by manual means. The scraper shown in the accompanying drawing has been used successfully in one plant to correct this condition, and it can be quickly and easily constructed and installed by an ordinary mechanic.

For a feeder four feet wide the scraper is constructed of one piece of 3-in. by $\frac{1}{4}$-in. iron bent as shown in section *A-A* and is hinged at points *B* so that the scraping edge *C* cleans off each pan as it comes around. The spring *S* causes the scraping edge to bear firmly against the pans and to cut all adhering material loose so that it falls into a bypass chute and thence to the crusher discharge. It will be noted that the side arms *D* cause the scraper to snap from one pan to the next and at the same time prevent the scraping edge from becoming caught in the return run of the feeder or in the depressions between the pans. The arrangement should be mounted so that the horizontal part which does the scraping is parallel to the face of the individual pans, so that no material may become caked in the depressions between the pans.

Electrical Timing Device for Ore Samplers

At the Garfield mills of the Utah Copper Co. an electrical timing device, designed by the engineering staff and made in the shops of the company, is in use for the operation of all of the motor-driven samplers in each mill. All of the samplers cut their respective ore streams simultaneously and at regular intervals. This is effected by the master timing device shown in Fig. 1.



12-11-26 ①

Fig. 1—Master timing control for automatic samplers

The device is continuously driven by a ⅛-hp. motor operating at 1,200 r.p.m. The motor revolutions are reduced by two worm-gear drives in series. On the shaft of the second worm gear is a small crank which operates a pawl engaging in a toothed wheel. Each revolution of the crank advances the toothed wheel an angular distance equal to the space of one tooth. The shaft of the toothed wheel carries upon its extremity a pair of electrical make-and-break contacts at opposite points. These are inclosed in a case provided with a glass front. In each revolution there are two contact intervals of a definite time length. The electrical current operates the motors, driving each sample cutter

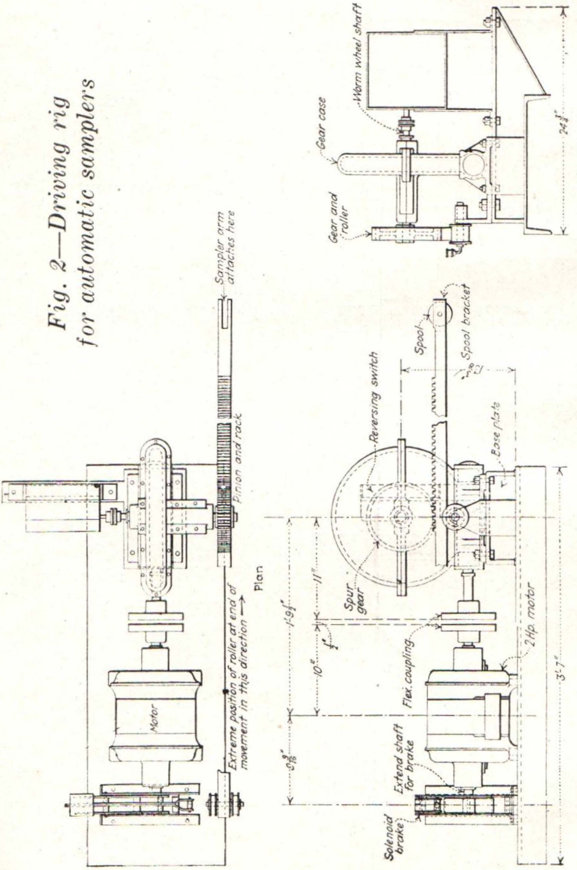


Fig. 2—Driving rig for automatic samplers

through relays which open and close the motor circuits automatically.

Each sample cutter is operated by 2-hp. induction motor, shown in Fig. 2, which drives a worm gear, a pinion on the shaft operating a toothed rack on rollers, to the end of which is attached the bucket arm of the sampler. The length of the rack is proportioned to the width of stream to be cut. A reversing switch returns the rack to its original position after making its outward and inward cut of the pulp stream. The rig is compact and operates satisfactorily.

Cananea Smelter Improves Reverberatory Practice

BY A. D. WILKINSON
Smelter Superintendent, Cananea, Sonora, Mexico

THE PRACTICE of smelting the entire charge in the reverberatory furnaces without roasting or drying has now been in operation for a year at the Cananea smelter of the Cananea Consolidated Copper Co., S. A., and sufficient data obtained to justify the statement that, under the operating conditions at Cananea, such a procedure is successful and economical.

The smelting plant consists of twelve roasters, two reverberatory furnaces, eight blast furnaces, and six converters; with a small crushing plant capable of reducing the reverberatory charge to a maximum of ⅜ in., and three mixing and storage beds operated on the Messiter system. Prior to the use of differential flotation at the Cananea concentrator, the smelting plant had been operated at a maximum capacity at different periods.

The operation of differential flotation on Cananea sulphide ores resulted in a material reduction of concentrate tonnage, and an increase in the copper content of the concentrates from about 4½ to 17 per cent copper. A change in the physical character of the concentrates was also effected, the colloid content being reduced so that the concentrates were drier and capable of being handled through bins and chutes as readily as fine ore. The effect of this change at the smelter was a lower smelting cost per pound of copper and increased recovery due to a smaller tonnage of slag and fewer metallurgical units in operation, only six roasters, one reverberatory, and two converters being necessary.

The roaster plant was not equipped with a positive dust collecting system, having only a dust chamber large enough to secure low velocity of the gas flow. This, though adequate for comparatively coarse low-grade feed, was inadequate for the collection of dust from the differential flotation concentrates. Various attempts were made to avoid this heavy loss. By reason of the low sulphur content of the roaster feed, it was possible to force the tonnage to be roasted through three roasters instead of six, care being taken to keep the amount of excess air at a minimum. This improved conditions, but the stack loss was still too high. The concentrates were then taken out of the roaster feed and charged wet to the reverberatory. This caused an unsatisfactory feed to the roasters and an unbalanced charge to the reverberatory, with unsatisfactory results. An attempt was then made to partially dry the entire charge in the roasters, which resulted in the sulphides melting out in the reverberatories, leaving a highly siliceous material that fused slowly and soon choked the furnace.

It was then decided to smelt the entire charge without roasting or drying. This was found to reduce the capacity of the reverberatory furnace about 50 per cent, but the results were so satisfactory that another reverberatory was started up to take care of the tonnage, and the roaster plant operation discontinued. The following gives the major items of interest in connection with the present Cananea smelting practice:

1. An increased recovery of 9.4 lb. of copper was made per ton of dry new material treated, of the analysis given below.

2. The cost of installation and operation of a mechanical dust-catching system is saved, with a much greater recovery than such a system could give.

3. The reduction in copper losses comes from eliminating roaster stack losses, eliminating dust losses from handling and charging the hot fine calcine and dust, and from the lower copper content of the slag.

4. The effect of the moisture content of the charge

in the reverberatory is to facilitate the elimination of sulphur—probably by retarding the melting of the sulphides, although there may be some reaction at the high temperatures of the smelting zone. The moisture also tends to eliminate dusting inside the furnace, thus reducing the fluxing action on the silica brick and the amount of dust carried out to the flues and boilers and on out the reverberatory stack.

5. Better advantage can be taken of the heat from the burning of oil and oxidation of elements of the charge itself than when the roasters are operated.

The new material in the reverberatory charge carries 10.6 per cent moisture and its analysis is 8.71 per cent Cu; 18.3 S; 25.2 SiO_2; 8.4 Al_2O_3; 20.9 Fe; and 1.8 per cent CaO. It has amounted to 91.88 per cent of the dry solid charge, the balance being cold converter slag shells. The molten converter slag poured back into the reverberatories has been 30.71 per cent of the solid charge. About 10 per cent of the new material received for treatment is utilized as silica for converter flux.

The matte has run 22.45 per cent copper, and the analysis of the slag has averaged 0.25 per cent Cu; 40.1 SiO_2; 12.4 Al_2O_3; 35.8 FeO; and 2.6 per cent CaO.

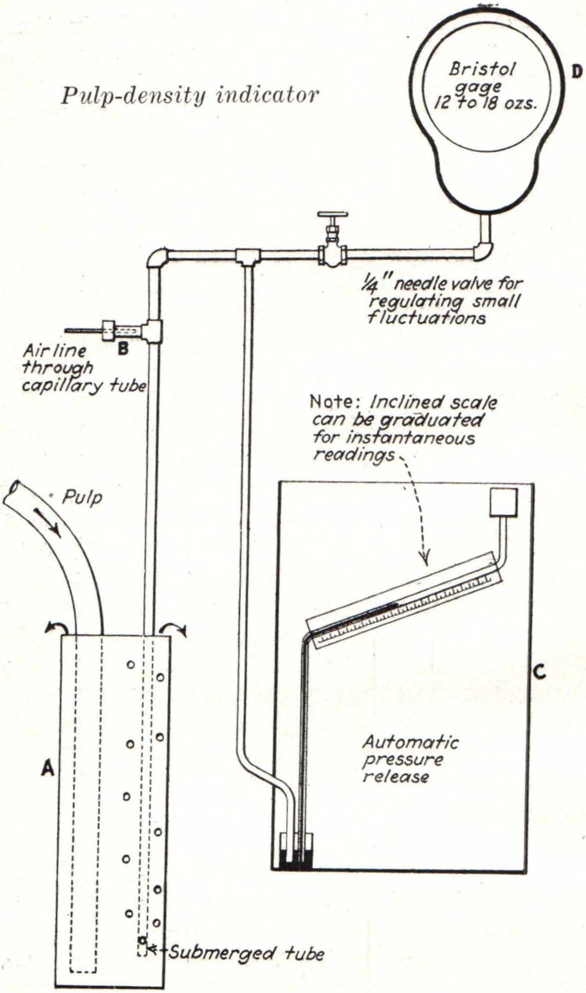
The gross ratio of barrels of oil per dry ton of solid charge has been 1.17. Of the sulphur charged, 37.54 per cent was eliminated in the reverberatory. The copper recovery was raised by 5.70 per cent and the silver by 4.25 per cent.

Pulp-Density Indicator and Recorder

By H. R. Banks and C. A. Remington
Metallurgist and Testing Engineer, Sullivan Concentrator, Kimberley, B. C.

Recently there has been developed at the Sullivan concentrator of the Consolidated Mining & Smelting Co. of Canada, a pulp-density indicator and recorder which is proving very useful in flotation control. By means of the inclined scale, shown in the accompanying illustration, instantaneous readings can be taken and the Bristol recording pressure gage provides a chart showing the variations in density of flotation feed pulp throughout the twenty-four hours. The instrument can be arranged with the inclined scale for the use of the

Pulp-density indicator


Bristol gage 12 to 18 ozs.
D
¼" needle valve for regulating small fluctuations
Air line through capillary tube
B
Note: Inclined scale can be graduated for instantaneous readings
Pulp
C
Automatic pressure release
A
Submerged tube

operators, and the recording device placed at some point distant from the machines, where it will be convenient for the use of foreman or superintendent.

The present instrument is based on a type of density indicator proposed by Prof. H. E. T. Haultain, of the University of Toronto. The accompanying figure will illustrate the principles employed.

In the illustration, *A* represents the pulp receptacle with pulp entering through the curved pipe and continuously overflowing. It is a 6-in. pipe, 20 in. long A submerged tube extends to approximately 18 in. below the overflow lip of the pulp receptacle. At *B* there is a capillary tube which allows air to enter the submerged tube, and this air bubbles out from the bottom of it through four $\frac{1}{16}$-in. holes drilled into the pipe.

A Bristol recording pressure gage, Model 11, is represented at *D*. This has a pressure range of from 12 to 18 oz.

An automatic pressure release is indicated at *C* to protect the pressure gage in the event of a choke-up causing air pressure to rise above the range of the gage. Pressure recording is that required to overcome the pulp head equal to the submergence of the tube plus the friction in the air line. The gage has been calibrated empirically to take care of air line friction and other small factors, and the recorded densities check the densities obtained from actual weighings. The 12-oz. dead pressure is the approximate equivalent of the submergence.

Specific Gravity Determination for Solids

By W. H. Seamon

Professor of Mining and Geology, College of Mines, University of Texas, El Paso, Tex.

The specific gravity of any solid is found by dividing a given weight of the solid by the weight of an equal volume of water, under standard conditions. The usual method employed requires a balance and a specially designed apparatus. The equipment is relatively expensive and considerable time is required. For some time I have employed a 10 c.c. graduated glass cylinder and a cheap balance, all of which could be carried safely in the pocket. It has proved satisfactory for quick field tests. The method of use is as follows: Break the solid to fragments, none larger than one-fourth of an inch; weigh quickly from 4 to 5 grams. Fill the cylinder up to the 5 c.c. mark with any liquid that will not act on the solid; introduce the weighed solid into the liquid quickly, tapping to release gas bubbles and read the increase of volume of the liquid as accurately as the graduations and the eye permit. The increase in volume, expressed in cubic centimeters, divided into the weight in grams is the specific gravity.

The following table gives the results I have obtained by the old and the method just described:

Mineral	**Old Method**	**New Method**
Graphite	2.15	2.06
Iron	7.3	7.17
Realgar	3.5	3.54
Molybdenite	4.7	4.64
Sphalerite	4.11	4.17
Pyrite	4.96	4.76
Anorthite	2.75	2.64

The method can be, and has been, employed in mills and other places where 50 to 100 grams of solid is conveniently available by pouring 100 c.c. of fluid into a graduated cylinder that will measure up to 150 c.c., adding the solids, shaking to displace air, and noting the increase in volume.

Treatment of Roaster Flue Dust

By G. Newton Kirsebom
90 West St., New York City

FLUE DUST, especially when it is of a complex nature, offers many difficulties to the smelter metallurgist. It is one of the undesirable byproducts that is too valuable to throw away and yet often of such physical and chemical characteristics that it is difficult to find a place for it in the smelting charge, so it is sometimes merely stocked in the hope that a profitable means of utilizing it may be developed with changed conditions. Such a stock of about 20,000 tons of baghouse dust accumulated at a plant in which I was interested, that was closed down owing to mine exhaustion. It was essentially a zinc material, for the treatment of which the company had put up a small experimental plant that had given the operators some experience in the electrolysis of zinc. Its analysis was approximately as follows: Lead, 7 per cent; zinc, 14; bismuth and copper each 0.5 per cent; arsenic, 8 per cent; and silver, 10 oz. per ton.

The lead was present largely in the form of sulphate and the zinc as a basic sulphate, so a wet method of treatment was indicated. Several methods were tried, of which leaching with roaster gas seemed the most promising. Mixing the dust with weak sulphuric acid, and passing a gas containing from 1 to 2 per cent of sulphur dioxide through the pulp, resulted in the solution of practically all of the zinc that was in the form of sulphate. The sulphuric acid alone did not give nearly so good results, indicating that the sulphur dioxide was the important factor in the solution. It was possible, by using small amounts of solution, to bring the zinc sulphate up to a concentration suitable for electrolysis. Unfortunately, all the arsenic went in solution, along with smaller amounts of iron, a condition that proved first to be a great stumbling block. To oxidize all the arsenic by artificial oxidizers, such as permanganate, was far too expensive, and any other way of separating the two metals seemed at first impossible.

Finally it was found that by adding a solution of barium sulphide slowly to the solution it was possible to precipitate out all the arsenic as sulphide, along with some barium sulphate, leaving the zinc sulphate practically untouched. The heavy precipitate of arsenic trisulphide and barium sulphate could now be mixed with powdered coal and heated in a furnace, from which fume of arsenic trioxide was recovered, at the same time as the barium sulphate was reduced to barium sulphide, which could be used over again in the process. The remaining zinc sulphate was then purified by usual methods and electrolyzed. Barren electrolyte was used partly for repeated leachings, along with the sulphur dioxide gas, and partly as acidifier in the subsequent brine process.

The residue was now leached with hot acidified brine, throwing the lead, bismuth, copper, and silver into solution as chlorides. The lead chloride was precipitated by cooling and treated separately, being melted down with lime and pulverized coal, giving pure lead and a slag of calcium chloride which was used for purifying the brine solution in the cycle. All of the other metals in the solution were precipitated with sponge iron, giving a metallic residue of lead, silver, copper, and bismuth, which could then be treated to recover the various metals. The residue after the brine leach was worthless and could be thrown away.

The process looked profitable on paper under the existing conditions, and it was practically decided to go ahead, when the slump in arsenic set in. Even so, the process might have been profitable had it not been for the fact that only 20,000 tons was available, so that it would not be possible to write off the invested capital by the time the material would be exhausted. Under the conditions, the scheme was abandoned and a good opportunity was lost for trying out a wet treatment of a complicated flue dust on a working scale. This has not been done so far in this country and for several reasons.

To the smelter superintendent, flue dust is an obnoxious and unpleasant byproduct, an economic burden, for the treatment of which he does not like to place heavy investments, and the result is that each plant treats its flue dust in a humdrum way. This is particularly the case with flue dusts from roasting and sintering furnaces, baghouses, or Cottrells. In some places the dust is stirred up with water, allowed to dry up and harden in irregular lumps, and then mixed with the blast-furnace charge. At other plants the dust is wetted with water and compressed in a briquetting machine, the briquets being allowed to harden in the air for several days. In both operations the labor expense is heavy. Briquetting of the dust costs anywhere from 80c. to $1.20 per ton, this being mostly for labor. With the increase in roaster dust, due to the changed conditions resulting from selective flotation, this expense will no doubt rise considerably in the future unless an adequate method of treating the dust is discovered.

As is a natural to a smelter man, any wet process is rarely considered. A wet process installed in connection with a smelting plant calls for men trained in hydro-metallurgy, and a smelter superintendent, being used to large and quick returns, looks with impatience on a process in which the investment is large compared to the daily output.

Most of the experimental work carried out at various plants has been in an effort to improve the mechanical treatment of the dust. To eliminate labor was the chief aim; tests have been carried out from time to time in which the labor in briquetting was supplanted by mechanical apparatus. "Nodulizing" the dust has been tried several times. It was wetted in a mixer, mechanically charged into a revolving furnace or drum which was slightly heated inside from a coal fire below, and small nodules of the dust would then come out from the lower end of the furnace. The principle is the same as in briquetting, except that the method described is supplanted by the rolling action of the furnace, heat being at the same time supplied to cause the nudules to dry up and harden quickly. Nodules could usually be made, suitable for charging into the blast furnace, but the percentage of loose dust was considerable and there was also trouble in keeping the furnace from clogging. Additions of small amounts of lime or sodium carbonate to the dust have, in certain cases, proved beneficial in giving larger nodules and lowering the amount of dust formed. In carrying out such a test I also found that the nodules coming out of the furnace wet would dry and harden very quickly by being subjected to a rolling action on inclines before reaching the bin.

RESEARCH ON DUST CONSTITUENTS SHOULD PRECEDE EXPERIMENTS

An intelligent research in the determination of the chief constituents of a certain dust and their relation to one another, and then on the effect of adding various amounts of cheap ingredients that would tend to ball the dust together and cause a quick hardening and drying of the product, would be constructive as a guide in such experiments. There is hardly any doubt that mixtures could be found to suit the different conditions, and the mechanical difficulties should not be insurmountable, considering the saving in cost. Such nodulizing, even with the use of small amounts of sodium carbonate, could be carried out easily for less than half the cost of briquetting.

Considering now the wet treatment of roaster flue dust, it is probable that this is the road along which the final solution to the question will be found. So far, experiments have been carried out only in a small way. We know that most of the dust produced in lead smelters carries more or less zinc and arsenic besides lead and previous metals. We also know that by making a wet pulp of the dust, and passing roaster gas through it, it is possible to dissolve out the first two elements mentioned. The zinc content will in the future be insignificant and scarcely worthy of recovery. But by applying the barium sulphide process to the solution obtained, and roasting the resultant precipitate, a refined arsenic can be produced at once with only the extra expense of reducing the barium sulphate to sulphide with pulverized coal. A short cut is thus made in the recovery of arsenic at the smelter, saving the blast furnace from this unpleasant element, obviating the formation of speiss, and giving a blast-furnace dust high in cadmium if this metal be present. It will, of course, also reduce the amount of blast-furnace dust to be treated and eliminate entirely the costly process of refining arsenic from crude.

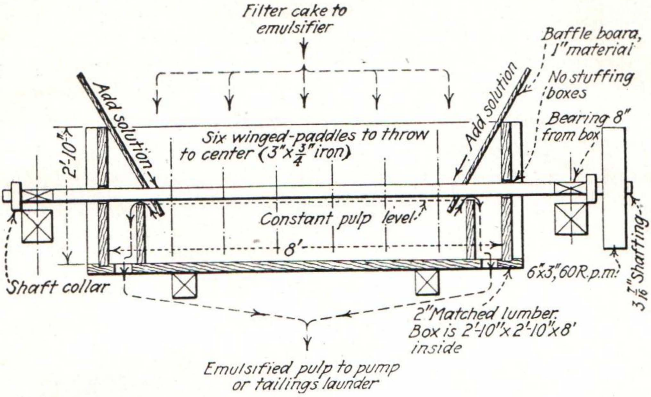
The residue from the sulphur dioxide leach could be properly treated by subjecting it to a brine leach. There has been much unpleasant experience where ores have been treated by a brine leach, especially on account of the excessive deterioration of equipment. However, flue dust, being a byproduct and in a fine state of division, lends itself more readily to brine treatment than any oxidized ore, which must undergo several stages of preparation. As to the materials used for containers, the good results obtained in the use of sulphurized concrete and similar material produced by the sulphur companies in the South should offer a solution to this feature of the problem.

Repulping Box for Handling Filter Cakes

BY ROBERT E. DYE

Timmins, Ontario

When double filtration was adopted at the cyanide plant of the Vipond Consolidated Mines, Ltd., Timmins, Ont., some months ago, it was necessary to devise a ready means for repulping or "emulsifying" the filter cake between stages as well as before pumping it to waste. The accompanying illustration shows the repulper which was built. Two of these are in use, each of which repulps 300 tons, dry weight, of filter cake per twenty-four hours with an equal weight of solution or water, giving a product which is absolutely


Filter cake to emulsifier
Baffle boara, 1" material
No stuffing boxes
Add solution
Add solution
Bearing 8" from box
Six winged-paddles to throw to center (3"x$\frac{3}{4}$" iron)
2'-10"
Constant pulp level
8'
Shaft collar
6"x3", 60R.p.m.
$3\frac{7}{16}$" Shafting
2" Matched lumber. Box is 2'-10"x2'-10"x8' inside
Emulsified pulp to pump or tailings launder

Emulsifier used at Vipond Consolidated Mines, Ltd.. Timmins, Ontario

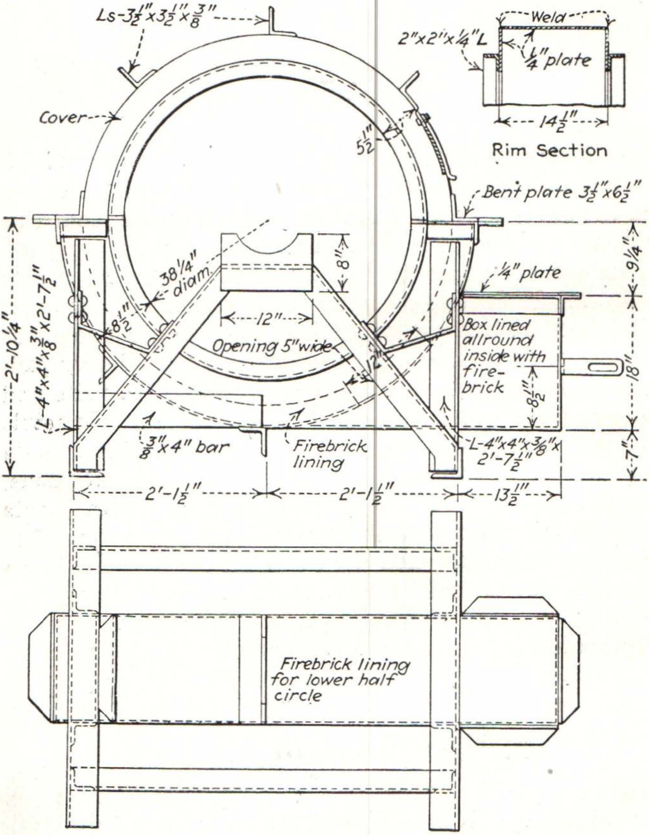
free from any lumps of filter cake not perfectly pulped. About one horsepower is required for each machine. In the six months of operation since the filter plant was started the machines have given no trouble.

In the design of the machine, the following points were established as essential to the success of the unit; (1) Stuffing boxes were avoided, as these are a constant source of annoyance; (2) a large shaft and heavy paddles were provided to secure rigidity and long life and to eliminate bearing trouble; (3) sufficient volumetric capacity was provided so that an appreciable time would be given to the operation, even though it is continuous, thus reducing the required speed of the revolving parts and at the same time the power and the wear. The machine was designed to meet requirements at the Vipond, but it is felt that the ideas involved could be readily and advantageously adapted to the design of repulpers for the same or similar conditions.

Roll Shell Heating Device

At the Garfield mills of the Utah Copper Co. worn roll shells are stripped by cutting with an oxy-acetylene torch. New shells are placed upon the rolls and are heated in a special oil-fired furnace which receives its oil supply from a portable pressure tank, compressed air being used for the operation of the burner.

The furnace which was first used at the Alaska Gastineau mill was introduced into the Arthur plant by Roy Hatch, the present superintendent. It is in two parts. The lower part contains the burner and combustion chamber and the upper part is removable. The lower part is provided with angle-iron frames to support the projecting ends of the shaft, and is lined with firebrick. In operation, the roll is picked up in a sling and placed in the lower part of the furnace. The burner is started and the cover placed in position. More



Sketch of roll shell heating furnace

than half an hour is required to heat the shell when the top part of the furnace is removed by the crane and the roll lifted out and set on floor jacks, the follower plates of the roll being set in position by a battering ram when they are bolted up. About an hour or less is required to set a new roll shell in place. The accompanying illustration shows the design of the furnace.

Obviating the Harmful Effect of Soluble Salts in Flotation

BY A. W. HAHN
Manager, Eureka Metallurgical Co.,
Salt Lake City, Utah

THAT THE PRESENCE of soluble salts has a marked influence on the flotation of various ores is a well-established fact. The usual soluble salts present in flotation pulps, due to the solution of the inherent salts in the ore or those in the water used, are ferrous and ferric sulphates, manganous and manganic sulphates, aluminum sulphate, sodium sulphate and chloride, and lime and magnesium salts. The influence of the salts of iron and aluminum, and methods for eliminating these effects in the flotation of copper ore have been the subject of an investigation to be summarized here. Further work on lead and lead-zinc ores is contemplated.

Through the courtesy of the Utah Copper Co., a quantity of ore was obtained containing a minimum of non-sulphide copper. This assayed 0.98 per cent total copper, of which 0.03 per cent was non-sulphide copper, the amount of soluble salts in this ore being negligible. Flotation tests were carried on in a laboratory Janney flotation machine. The flotation reagents used were xanthate and Cleveland Cliffs creosote No. 2, these reagents being used in the same quantities in all tests. Chemically pure burnt lime and ferrous, ferric, and aluminum sulphates were also used.

Table I shows the effect of the addition of various amounts of ferrous, ferric, and aluminum sulphates, keeping the lime a constant. Table II shows the effect of increasing the amounts of lime in conjunction with increasing amounts of soluble salts.

Table I—Effect of Addition Agents on Grade of Tailing, Keeping Lime Constant

Reagents: 4 lb. CaO, 0.2 lb. xanthate, 0.5 lb. C.C. No. 2, per ton of ore

Ferrous Sulphate		Ferric Sulphate		Aluminum Sulphate	
Lb. $FeSO_4$ Per Ton Ore	Tailing Per Cent Copper	Lb. $Fe_2(SO_4)_3$ Per Ton Ore	Tailing Per Cent Copper	$Al_2(SO_4)_3$ Lb. Per Ton Ore	Tailing, Per Cent Copper
0.0	0.07	0.0	0.07	0.0	0.07
1.5	0.19	2.0	0.21	2.0	0.10
3.0	0.16	4.0	0.19	4.0	0.12
4.5	0.22	6.0	0.46	6.0	0.17
6.0	0.79	8.0	0.44	8.0	0.21
7.5	0.73	10.0	0.32	10.0	0.42

Table II—Effect of Increased Lime and Soluble Salts on Grade of Tailing

Reagents: X lb. CaO, 0.2 lb. xanthate, 0.5 lb. C.C. No. 2, per ton of ore

Ferrous Sulphate			Ferric Sulphate			Aluminum Sulphate		
Lb. CaO Per Ton Ore	Lb. $FeSO_4$ Per Ton Ore	Tailing Per Cent Cu	Lb. CaO Per Ton Ore	Lb. $Fe_2(SO_4)_3$ Per Ton Ore	Tailing Per Cent Cu	Lb. CaO Per Ton Ore	Lb. $Al_2(SO_4)_3$ Per Ton Ore	Tailing Per Cent Cu
4.0	0.0	0.07	4.0	0.0	0.07	4.0	0.0	0.07
5.0	1.5	0.10	5.0	2.0	0.08	5.0	2.0	0.10
6.0	3.0	0.13	6.0	4.0	0.10	6.0	4.0	0.12
7.0	4.5	0.14	7.0	6.0	0.13	7.0	6.0	0.12
8.0	6.0	0.17	8.0	8.0	0.11	8.0	8.0	0.09
9.0	7.5	0.23	9.0	10.0	0.14	9.0	10.0	0.10

Table III—Effect of Oxidizing Agents on Grade of Tailing

Test	Lime Lb. Per Ton	Bleach Lb. Per Ton	Xanthate Lb. Per Ton	Frother Lb. Per Ton	Tailing Per Cent Cu	Feed Per Cent Cu
Utah Copper ore and 7.5 lb. $FeSO_4$ No. 1	4.0	0.0	0.10	0.50	0.17	0.98
No. 2	4.0	2.0	0.10	0.50	0.12	0.98
No. 3	4.0	3.0	0.10	0.50	0.09	0.98
Braden Copper Co. ore, No. 1	4.0	0.0	0.20	0.50	0.58	2.34
No. 2	4.0	1.0	0.20	0.50	0.42	2.34
No. 3	4.0	2.0	0.20	0.50	0.44	2.34
No. 4	4.0	3.0	0.20	0.50	0.45	2.34
No. 5	4.0	4.0	0.20	0.50	0.45	2.34
No. 6	4.0	0.0	0.20	0.50	0.35	2.12
No. 7	4.0	2.0	0.20	0.50	0.26	2.12
No. 8	4.0	*(a)*	0.20	0.50	0.26	2.12
Chino Copper Co. ore No. 1	4.0	0.0	0.10	0.50	0.35	1.53
No. 2	4.0	2.0	0.10	0.50	0.25	1.53

(a) Oxygen used instead of bleaching powder.

It is evident from Table I that ferrous sulphate, even when present in small quantities, is extremely detrimental and affects copper recoveries to a marked degree. Ferric sulphate is also detrimental, but its effect appears to reach a peak, beyond which copper recoveries are not decreased. Aluminum sulphate is also detrimental, although to a lesser extent than are the iron salts. Table II shows that the addition of burnt lime to a large extent counteracts the harmful effects of ferric and aluminum sulphates; but ferrous sulphate, even when excessive amounts of lime are present, still adversely affects copper recoveries.

With these facts established, it was logical to try next the effect of adding an oxidizing agent and lime to the ore pulp, and then subject this treated pulp to flotation. Experiments were carried out on Utah Copper Co. ore to which ferrous sulphate had been added, and also on samples of ore from the Braden and Chino Copper companies, which contain soluble iron salts. Chloride of lime and oxygen gas were used. The chloride of lime was, with burnt lime, added at the ball mill and the oxygen gas was bubbled through the ore pulp in the presence of lime. Flotation reagents were added in the flotation machine after the oxidizing treatment. Table III shows the tailings obtained, both with and without this treatment.

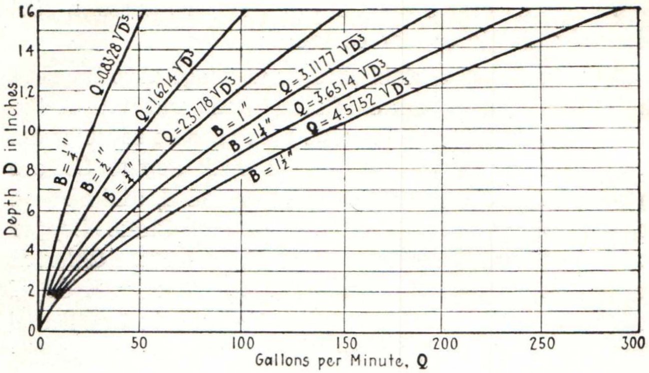
Table III shows that by the addition of 2 lb. of bleaching powder, from 1½ to 3 lb. of copper can be saved per ton of tailing produced. The results using oxygen instead of chlorine (bleach) are the same, but bleaching powder is easily measured and controlled. An excess of oxidizing agent is to be avoided, not only on account of its cost, but an excess is detrimental to copper recoveries, probably because of the destruction of organic flotation reagents. The odor of chlorine should be absent when the pulp goes to the flotation machine, and before flotation reagents are added.

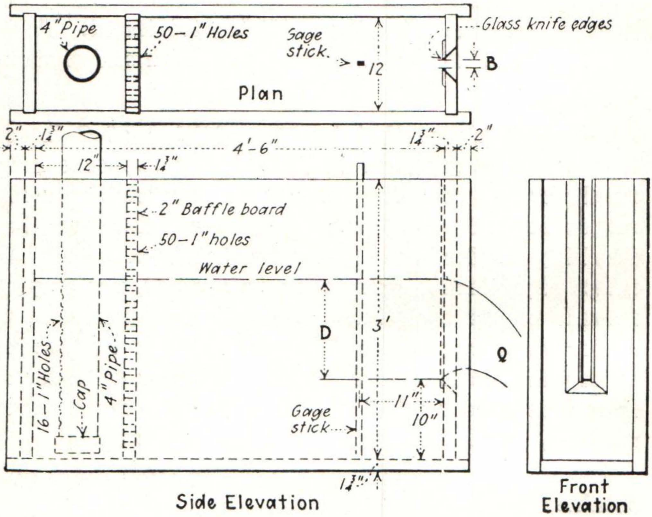
The following deductions are drawn from the results obtained:

1. Ferrous, ferric, and aluminum sulphates seriously affect copper recoveries, even when present in relatively small amounts.

2. Burnt lime eliminates, to a large extent, the deleterious effects of ferric and aluminum sulphates, but offsets to only a small extent the effect of ferrous sulphate.

3. By adding an oxidizing agent, such as oxygen, chlorine, or chlorine of lime, to the ore pulp in the presence of burnt lime, the effect of ferrous sulphate can be almost completely eliminated.





Flow of Water Through Narrow Weirs

By Harold A. Linke

In mill operations, especially experimental work, it is frequently highly desirable to know the quantity of water added at various points. The accompanying figure shows a simple weir box for the purpose of measuring water flow. Curves and their respective formulas for the discharge of weirs of the following widths: ¼ in., ½ in., ¾ in., 1 in., 1¼ in., and 1½ in., are also shown.

Use of Portable Protecting Screens

Accidents, more especially eye accidents, arise where metal chipping and hammer work are in progress. To guard against flying chips and bolt heads, portable screens are in use in the Garfield mills of the Utah Copper Co. These screens are about 6 ft. high, each screen being formed of three hinged divisions so that the work can be surrounded on three sides. The upper part of each division of the screen is formed of fine mesh wire cloth extending from the top of the screens frame to a short distance below the top of the working bench. The arrangement effectively stops all flying particles.

In the electric welding shop a similar screen is used to inclose the work. A black oilcloth is used on the screen compartments and extends from the top to within a short distance of the floor.

Determination of Zinc in Roasted Ore

BY JOHN H. HASTINGS

Chief Chemist, American Steel & Wire Co. Zinc Plant, Donora, Pa.

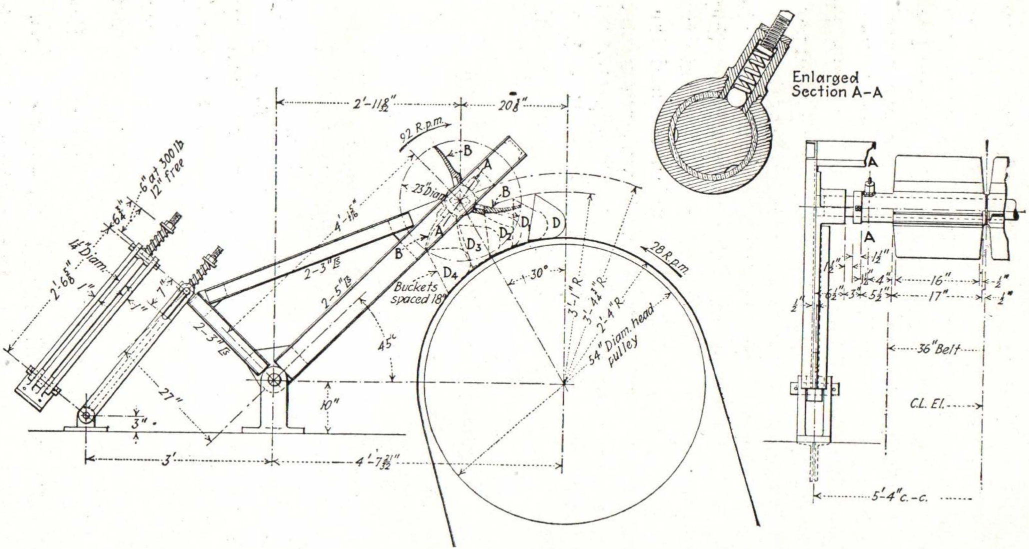
To ascertain the content of zinc in roasted ore, one-half gram of ore is weighed into a 250 c.c. beaker. The sample is wet down with water from a wash bottle and 15 c.c. hydrochloric acid added. The acid is evaporated to about 5 c.c. on the hot plate near the boiling point, but not boiled. Care is taken that it is not overheated, as zinc chloride is volatile above 125 deg. C. Seven grams of ammonium chloride and 200 c.c. of ammonia are added, and the beaker is filled half full of hot water. The solution is boiled five minutes and filtered through a No. 2 Whatman filter paper into a 400-c.c. beaker. The iron precipitate is washed ten times with a solution containing 100 gm. of ammonium chloride and 50 c.c. ammonia to the liter of water. A liter serves for twelve determinations. The solution is neutralized, using methyl orange as an indicator, with hydrochloric acid and 5 c.c. hydrochloric acid added in excess. The methyl orange color is taken out with a drop or two of bromine water. It is then boiled gently for fifteen minutes to get rid of oxydizing agents and titrated with potassium ferrocyanide, 1 c.c. of which is equal to 0.005 gram of zinc. A 5-per-cent solution of uranium nitrate is used as an indicator on a paraffine spot plate. The end point is the first change of color to pink. Before titrating, 50 c.c. of a solution containing 10 gm. of sodium hyposulphite to the liter of cold water are added to the zinc solution. The solution is stirred vigorously and titrated slowly. The method follows the methods given in Low's "Technical Methods of Ore Analysis."

The potassium ferrocyanide solution is standardized by weighing 0.350 gm. of c.p. zinc into a 400-c.c. beaker. Ten cubic centimeters of water and 6 c.c. of hydrochloric acid are added, and the zinc is dissolved. Twenty-one grams of ammonium chloride are added, and the solution is made up to 250 c.c. with hot water, after which it is boiled for fifteen minutes and titrated as for ores. The same condition as to amount of ammonium chloride in solution, acidity, and the temperature and volume is maintained in the standard as in the ore determination. The standards are run in duplicates. The raw-ore samples are analyzed the same as the roasted ore, with the exception that the sample is carried to complete dryness with 5 gm. of potassium chlorate and 10 c.c. of nitric acid. If manganese is present it is precipitated by the addition of 1 gm. ammonium persulphate with the ammonia, or 10 to 25 c.c. bromine water. When copper is present a 1 per cent solution of sodium sulphide is used instead of the hyposulphite.

Cleaning Device for Elevator Bucket

An ingenious device was recently designed by the engineering staff of the Utah Copper Co. for cleaning elevator buckets when charged with wet and sticky ore. Under such conditions the buckets do not completely discharge and the accretion tends to build up, interfering with the proper operation of the bucket elevator. The device consists of a revolving scraper wheel mounted upon a triangular-shaped frame, shown in the accompanying illustration. A certain amount of

Suggested bucket-elevator cleaning device



play about a fixed radius is allowed, but the scraper wheel frame cannot interfere with the elevator-bucket clearance. To prevent spinning of the scraper wheel, a spring-actuated brake, shown in the enlarged section *A-A,* is attached to the hub of the scraper wheel.

The scrapers B revolve and are so timed that they engage the elevator bucket D at D_1. As the bucket travels forward, scraper B moves along the upper side of the bucket to the bottom, as in D_2, and by the time the bucket occupies the position of D_4, the scraper leaves the bucket. Traveling at different speeds, as they do, will necessitate exact construction and accurate timing.

A model of the device was constructed and the idea proved out on a small scale, but it was never installed in the milling plants at Garfield. The probability of its satisfactory operation is open to question, but it is presented here as an illustration of ingenious design.

Determination of Xanthate in Impure Solutions

BY A. F. HALLETT AND FRANK RYDER
Chief Chemist and First Assistant Chemist,
Inspiration Consolidated Copper Co.

THE SOLUTION whose analysis is to be discussed is a dark brownish-red liquid with a specific gravity of 1.212. It is the mother liquor produced during the manufacture of potassium xanthate by the alcoholic process, and is used as a flotation reagent.

In making this analysis three methods have been tried, all of which have been found to be of uncertain value. These methods are (a) the volumetric method using $\frac{N}{10}$ hydrochloric acid with methyl red indicator, (b) the direct precipitation of the xanthate with copper sulphate, and (c) the determination of the sulphur present, which is calculated to potassium xanthate.

(a) It is impossible to use a volumetric method, because the red solution masks the color of the indicator (methyl orange or methyl red), obscuring the end point.

(b) In determining potassium xanthate by precipitating with copper sulphate solution, the presence of carbonates, sulphides, and thiocarbonates will cause high results, as these compounds all precipitate copper from the copper sulphate solution in addition to that precipitated by the potassium xanthate.

(c) In the same way any method which determines the sulphur and calculates the potassium xanthate from it gives high results, as the sulphur in the potassium xanthate and that in the other sulphides present cannot be separated, but would be determined together.

The following method will determine potassium xanthate accurately in the presence of carbonates, sulphides, thiocarbonates, and carbon bisulphide, all of which are present in the solutions as impurities. The sulphides and thiocarbonates are removed and in the precipitation of the copper xanthate the carbonates and carbon bisulphide will not interfere. It depends on the fact that sulphides and thiocarbonates can be precipitated by an ammonical solution of zinc chloride and removed from the solution by filtering.

$$K_2S + ZnCl_2 = ZnS + 2KCl$$

$$K_2CS_3 + ZnCl_2 = ZnS + 2KCl + CS_2$$

The potassium xanthate in the filtrate can be precipitated by a copper sulphate-sodium tartrate solution and separated from the carbonates and carbon bisulphide by filtering.

$$2KCS_2(C_2H_5O) + CuSO_4 = Cu(CS_2 . C_2H_5O)_2 + K_2SO_4$$

To prove that a solution of ammoniacal zinc chloride will remove the sulphides and thiocarbonates, various samples of impure solutions were tested for these impurities before treatment with the ammoniacal zinc chloride. They were always found present in large amounts. After the precipitate produced by the ammoniacal zinc chloride had been filtered off, the filtrate was tested for sulphides and thiocarbonates; these were always found absent, indicating complete removal of sulphides and thiocarbonates. Further tests on pure potassium xanthate dissolved in water and treated with ammoniacal zinc chloride show that potassium xanthate is not affected by this reagent. Therefore, it can be assumed that the potassium xanthate in these solutions is not affected or precipitated by the ammoniacal zinc chloride and that the removal of the sulphides and thiocarbonates is exact and complete. That this operation is not variable, but is always complete, is further shown by the fact that duplicate assays checked each other within the limits of working error, and often exactly.

The tests used for sulphides and thiocarbonates are those contained in the article on "The Analysis of Xanthate," by Calcott, English, and Downing (*Mining Journal-Press,* Dec. 20, 1924). The method to be outlined is supplementary to the one there given.

Determine the specific gravity and temperature of the solution to be analyzed. (That the results obtained from day to day shall be comparative, the solutions must be at the same temperature. In our laboratory, 27 deg. C. is taken as the standard.)

Measure 10 c.c. of the solution and place in a 250-c.c. volumetric flask. Make up to the mark with cold distilled water and mix well by shaking. Place 25 c.c. of this solution in a 250-c.c. beaker and add 25 c.c. of distilled water. Add enough ammoniacal zinc chloride solution to precipitate the sulphides and thiocarbonates. About 10 to 15 c.c. of a 10 per cent solution, strongly ammoniacal, will be required. Place on the steam plate and warm five to ten minutes until the color changes from yellow to cream white, showing that the precipitation of the sulphides is complete. When it has settled fairly clear, filter and wash five times with dilute ammonia. To the filtrate (which should be about 300 c.c.) in a 500-c.c. Erlenmeyer flask, add 25 c.c. of a copper sulphate-sodium tartrate solution. (The copper sulphate-sodium tartrate solution is made by dissolving 25 gm. of $CuSO_4 . 5H_2O$ in 600 c.c. of hot distilled water. Cool to room temperature and add 200 gm. of c. p. sodium tartrate. After complete solution of the tartrate, slowly add 10 per cent Na_2CO_3 solution until the color of the reagent turns to dark blue and the reagent is just alkaline to litmus, indicating an excess of Na_2CO_3. Then dilute the solution to a volume of one liter and filter. This solution is the one used by the du Pont company.) At this point it should be at approximately room temperature or slightly above.

Place in a pan of cracked ice and water for one to three hours, with occasional agitation. Filter and wash with cold water until precipitate is free from copper sulphate. Ignite the precipitate of copper xanthate and place it, when cool, in a 300-c.c. electrolytic beaker. Add 5 c.c. HNO_3 and dilute to 25 c.c. Make it slightly ammoniacal, neutralize with HNO_3, and add 4 c.c.

excess. Add 3 c.c. H_2SO_4, dilute to 150 c.c., and electrolyze for forty minutes, using a rotating anode with 2 amperes and 125 volts for fifteen minutes and raising this to 4 amperes for the last twenty-five minutes.

From the weight of copper obtained, the potassium xanthate in the sample taken can be calculated as follows:

Weight of copper=0.0520 gm.

If 1 c.c. of the solution is taken and the specific gravity is 1.212, then

$$\frac{0.0520 \times 5.04}{1.212} \times 100 = 21.52 \text{ per cent potassium}$$

xanthate.

The factor 5.04 is obtained as follows:

$$2\ KCS_2\ (C_2H_5O) + Cu\ SO_4$$
$$= Cu\ (CS_2 .\ C_2H_5O)_2 + K_2SO_4$$

1 copper (63.5) reacts with 2 potassium xanthate (320.2)

$$\frac{320.2}{63.5} = 5.04$$

The following table shows the percentage of potassium xanthate before and after removal of the impurities in various impure solutions:

Sample Number	False Potassium Xanthate—Sulphides and Thiocarbonates Present		True Potassium Xanthate—Sulphides and Thiocarbonates Removed
	Potassium Xanthate, Per Cent	Specific Gravity	Potassium Xanthate Per Cent
A	34.5 34.3	1.206	22.5 22.3
B	40.4 40.6	1.212	16.3 16.2
C	42.7 42.4	1.206	33.1 32.9
D	45.9 45.9	1.202	21.4 21.4
E	44.0 43.7	1.213	33.6 33.9
F	40.1 40.1	1.195	31.8 32.0
G	45.8 45.9	1.212	37.4 37.4
H	48.6 48.4	1.232	38.1 38.0
I	46.0 46.0	1.213	39.9 39.6
J	24.8 24.9	1.112	18.6 18.6
K	12.8 12.8	1.058	9.3 9.1
L	12.3 12.4	1.060	9.5 9.6

The duplicate determinations of false and true potassium xanthate on each sample were made the same day.

Cyanide Precipitation Refinements

BY H. W. SHOEMAKER
Mill Superintendent, Lake Shore Mines, Ltd.,
Kirkland Lake, Ont.

In the precipitation of rich gold-bearing cyanide solutions, the reduction of the last 2 to 5c. per ton down to a trace of gold in the tailing requires, in addition to skilled operation, certain mechanical refinements that may be easily applied to the conventional Merrill-Crowe system. These are as follows:

The use of paper as a filtering medium is becoming more popular, with the manifest advantages of time saved in cleaning up and reclothing the precipitation press and lowered cost for supplies, as well as the fact that paper over 8- to 10-oz. duck is a more efficient filter than sheeting over canvas. But in such situations where the source of power for driving pumps is not dependable, or interruptions of solution flow that result through other causes are unavoidable, paper is not usually considered safe. This is because of the fact that after a sufficiently heavy cake of precipitate has collected on the paper, cessation of pumping allows the cake to fall and tears paper with it, leaving the bare canvas, which is an indifferent filter of gold slime unless heavily primed with zinc. This is much more noticeable with square-frame presses, such as the Shriver, than with the more commonly used triangular type, such as is usually supplied with a Merrill installation. If paper is to be used safely under such conditions, the remedy is to install a swing check valve in the solution main near the presses, and between it and the pumps a vacuum release of the ball valve type. This allows leakage back through the pumps when stopped, but holds solution in the press, thus supporting the paper and making priming unnecessary when pumping is resumed.

Reciprocating pumps are the most common for this service, and, for mechanical as well as metallurgical reasons, it is advisable to eliminate the pulsations of pressure in the solution line. This is usually solved by the use of air chambers at the pump discharge. But solutions already de-aërated by passage through the Crowe vacuum receiver are particularly lively in reabsorbing any stray air that may be brought in contact with them. Hence the air chamber is soon emptied of air, and its cushioning action ceases. The chamber may be recharged with air by connecting it with an air line having sufficient pressure to overcome the solution pressure at the pump discharge; but by so doing the deaërating process is in an equal degree nullified, and additional zinc is required to maintain efficiency. One method of eliminating this trouble is to interpose a flexible seal between the air in the air chamber and the solution it is expected to cushion. Another method is to substitute a spring alleviator for the air chamber. At the Lake Shore mill this condition is met by using a diaphragm-pump base connected to the solution line and substituting for the diaphragm a circle of heavy elevator belting having no opening. Above this is clamped a cover plate opening into the usual air chamber above. This air drum is provided with a gage glass, and, after assembly, enough water is poured into the drum and above the belting to fill the gage glass to a mark one inch above the lower fitting. The top of the drum is connected through a valve to high-pressure air for balancing pressure. When operating, the water level is an index of air required to balance the solution pressure. An adjustment once a day is all that is required, and there is no possibility of the air in the chamber reducing Crowe efficiency.

If the zinc feed is to be kept at a minimum it must be fed absolutely uniformly and in a fixed ratio with the solution flow. This necessitates accurate measurement of solution flow, and at this plant the use of a Venturi tube with recording and integrating device is satisfactory. It is also a factor in estimating accurately the gold to be expected at clean-up and is a check on theft. The ordinary zinc feeder does not feed continuously but in a series of dribbles. This intermittent feed is averaged up in the feed cone by the addition of a mechanical tapper to the zinc feeder. By thus cutting down the interval between portions of zinc dropped from the end of screw, a better average feed is assured. Any zinc dust will benefit and be of higher efficiency if tumbled in a miniature ball mill with an odd dozen of 2-in. steel balls. This treatment breaks up any agglomerates caused by dampness and rubs all the particles free of oxide and carbonate unavoidably present.

Unless lead be present in the ore under treatment, addition of small quantities of soluble lead salts is invariably beneficial in attaining best results. Through the use of lead acetate is well established by custom, the nitrate has several characteristics which make it preferable. It has a higher lead content, is purer as usually obtained commercially, and is more soluble. Its feed is small, about 0.01 oz. per ton of ore in practice at the Lake Shore mill, but, like the zinc, it should be fed as uniformly as possible. Portions of the daily feed, in concentrated solution, are added to a large diluting box fed by a small stream of pregnant solution and overflowing to the main storage tank. Under no circumstances is this concentrated solution to be fed at the vacuum seal cone with the zinc dust.

Most installations of the Crowe process include a single-cylinder vertical air-cooled vacuum pump, 6x4½ in., made by the Gardner Governor Co., of Quincy, Ill. In this service there is a marked tendency for the oil to be sucked by the piston and expelled in the exhaust to a degree that leaves the oil level too low unless frequent additions are supplied. After burning out several bearings, the small oil gage provided by the manufacturer was replaced with a sight oiler in which the oil level can be kept about three inches higher. In addition, the exhaust was piped directly back to the crankcase. Oil pumping now is a distinct advantage, as it seals the valves more effectually and is returned to the crankcase for re-use. Lubrication is assured by the higher oil level and the practical absence of leakage, since the actual amount of air displaced after the receiver is up to vacuum is small, and it escapes through the breather ports after dropping the oil.

Another improvement in the operation of this pump can be effected by giving the intake valve more clearance and a quicker return. The additional clearance is provided by machining the lower side of valve plate $\frac{1}{32}$ in. To make the valve return quickly to its seat, four of the holes through the valve plate were half plugged from the top and in the recesses thus made stiff coil springs were placed. These changes converted a mechanical nuisance, incapable of more than 20 in. of vacuum, into a smooth-running, foolproof machine pulling within 1 in. of the barometric maximum.

The present triplex plunger type of pump is soon to be replaced by two-stage centrifugal pumps with special seal arrangements. At the Lake Shore, the installation of the Crowe receiver is one of high head with respect to the pumps serving the precipitation press, so that there is no possibility of sucking air at the glands, but for installations of low head this type of pump would probably have to be run submerged to eliminate the possibility of sucking air.

The combination described in the foregoing has made it possible to precipitate gold solution of $8 to $10 per ton to a tailing solution of a trace to 1c. with a zinc consumption of 0.45 oz. per ton of solution and lead nitrate 0.002 oz. per ton. The resulting cost is $0.03 per ton ore for precipitation and $0.03 for the final refining, which is a direct smelt resulting in bullion of 850 total fineness.

Rapid Determination of Barium

By B. W. Deason
Salt Lake City, Utah

After titrating a lead solution with ammonium molybdate, put it back on the hot plate and boil for five minutes. If the ore contains barium it will require more molybdate to give an end point. If the ore contains much barium each repeated boiling will take more molybdate, sometimes raising the assay 5 or 6 per cent, but never raising it high enough to give a correct assay.

After much experimenting I devised the following method for ores containing barium, and it has been in use in the laboratory of Black & Deason for three years, having been also adopted as the regular method on this class of ore by most of the smelters and shippers in the vicinity of Salt Lake City.

Run the assay as usual until the lead sulphate is ready to be dissolved with ammonium acetate. Instead of adding ammonium acetate, add 5 c.c. sodium hydroxide solution (1 lb. NaOH to 1,000 c.c. water) diluted to about 25 c.c. with water. Boil and then dilute to the usual bulk for titrating. Make the solution acid with acetic acid, not more than 1 c.c. excess, using phenolphthalein indicator. Boil and titrate with ammonium molybdate, using tannic acid indicator. The sodium hydroxide must be added alone. When added in addition to ammonium acetate it has no effect.

Conveying Ore at the Ajo Plant

THE new concentrating plant of the New Cornelia Copper Co. is 5,000 tons' capacity, though at present it is running considerably over this. From the first, it has run smoothly and has been a credit to the skill of its designer, H. Kenyon Burch. The plant was built to treat a large body of sulphide copper ores found underlying the carbonate ores formerly treated in the leaching plant which occupied practically the same ground as the new concentrator. Construction was carried on in units. The first unit was erected and producing before the others were completed. This worked out well, and all the wrinkles were ironed out before the last units of the plant were completed.

The ore is mined by steam shovels using 4-cu.yd. dippers, and is hauled about a mile to the plant in 20-cu.yd. automatic dump cars. It is dumped into a concrete receiving bin holding 500 tons at the head of the primary crushing plant. Some of the ore runs very large. This ore is handled on a pan conveyor 8 ft. wide by 75 ft. long, at the rate of 10 ft. per minute, and is dumped over the primary crusher uniformly. The crusher reduces it to 6 in. size. This product is delivered by S-A belt conveyors traveling at the rate of 450 ft. per minute to an intermediate 6,000-ton steel storage bin. The belt bringing this material up is a 48-in. belt which discharges to a second 42-in. belt, which in turn deposits the material into the intermediate bin by means of a tripper.

The tripper is of steel-frame construction and of special design. At the bottom of this bin the ore is passed by apron feeders to four short 36-in. belt conveyors which travel at 100 ft. per minute. These conveyors deliver their product to four intermediate crushers which are protected by four 48-in. Burch grizzlies. After passing through these crushers and through the grizzly, the product is taken at the rate of 350 ft. per minute on four 24-in. belt conveyors from the intermediate crushing plant to crushers in the roll plant, which are again protected by 48-in. Burch grizzlies. The grizzlies above the crushers have a 2-in. opening. This material is brought down to 1¼ in. through the crushers, joins the by-passed undersize and goes onto the conveyors beneath and to 78x24 in. rolls.

This crushing plant, including both the intermediate crusher and the rolls, is divided into four units, which are entirely separate and fed from underneath the intermediate bin with separate feeders, the ore being carried from these feeders to the crushers and rolls on separate belt conveyors. From under the rolls the product is delivered by two long belt conveyors, one 48 in. wide and the other 42 in. wide, at speeds of 400 and 450 ft. per minute respectively, which carry the ⅝-in. to ¾-in. product to a 10,000-ton fine-ore bin at the head of the concentrator proper. This bin is of steel construction, parabolic in shape, and 300 ft. long. The product is discharged onto it by means of a belt conveyor tripper. The material is drawn from under this bin by manually operated gates onto 42-in. feeder belts running parallel with the length of the bin. Each of these belts is 26 ft. long and is driven at variable speeds, averaging around 30 ft. per minute.

The belts discharge to a single 24-in. conveyor which feeds the primary rod mills. From these mills the material passes to classifiers and from there to secondary rod mills and to a secondary classifier, and then to flotation machines. From these machines the concentrates are laundered to concentrate thickener tanks and then to the filter plant. A 24-in. belt conveyor 375 ft. long, traveling at 250 ft. per minute, discharges the concentrates from the filter plant directly into the concentrate car. The tailings are laundered from the flotation machines to concrete sand boxes from which the slimes overflow into a concrete slime thickener tank.

All crushing operations in the plant are protected by magnetic pulleys on belt conveyors in proper places, and in addition heavy magnets are hung over some of the belts as an added precaution. The output of the roll plant and of the grinding plant is recorded by means of weightometers used on the belt conveyors. In the grinding plant this record is kept by units, of which there are five, each extending at right angles to the large 10,000-ton fine-ore bin.

Precipitation and Refining at Tonopah Extension Mill

By H. A. Burk

Metallurgist, Tonopah Extension Mining Co., Tonopah, Nevada

The former method of precipitation of cyanide solutions at the Tonopah Extension Mill, Tonopah, Nevada, was by zinc shavings in zinc boxes, and the refinery practice consisted in melting the dried precipitate in Monarch-Rockwell furnaces. Consumption of zinc shavings was about 0.9 lb. per pound of gold-silver bullion recovered. The average grade of the dry precipitate was 65 per cent gold-silver, and the flux required to melt it was 10 per cent borax glass, 7.5 per cent soda ash, and 5 per cent silica sand. The ratio of silver to gold in the precipitate was about 100 to 1.

Improved methods of precipitation and refining were adopted about a year ago. Precipitation is now effected with zinc dust, and the precipitate is collected by the Merrill vacuum-leaf system. It is refined, without preliminary drying, by melting in a reverberatory furnace. Consumption of zinc dust is 0.64 lb. per pound of gold-silver bullion recovered. The average grade of precipitate is about 90 per cent gold-silver, and the flux required is 1.5 per cent borax glass, 1 per cent soda ash, and 2 per cent silica sand.

Under normal conditions, 500 tons of ore are treated per day, and about 1,500 tons of solution are precipitated. The solution consists entirely of the filtrate from the Butters filters, no circulating mill solution being precipitated. It is clarified by filtration through vacuum-leaf filters of the Butters type, and delivered into a storage tank 24 ft. in diameter by 7 ft. in depth.

The solution is then drawn continuously from this tank by a No. 4 Krogh centrifugal pump, submerged in solution to prevent access of air at the pump gland, and is discharged through two 3-in. float-regulated, Merco-Nordstrom valves into a two-compartment Merrill vacuum-leaf precipitating vat. This vat is constructed of reinforced concrete, each compartment measuring 6 ft. 8 in. in cross-section by 9 ft. 6 in. deep. In each compartment there are 18 vacuum leaves, 5 ft. high by 6 ft. long, suspended about 2 ft. below the top of the vat. These leaves are of somewhat different construction from the usual Butters type. They have the customary supporting framework of ¾-in. pipe, but are covered with burlap, instead of canvas, into which are sewed corrugated filter strips. Two thicknesses of light sheeting over the burlap complete the filter. The sheeting lasts about nine months.

Precipitation is with zinc dust, which is fed from slow-moving belt conveyors into auxiliary concrete compartments, integral with the precipitation vat. The zinc dust passes with the incoming solution into the leaf compartments near the bottom. Precipitate is kept in suspension in each compartment by means of a Devereau type of propeller, 24 in. in diameter, fixed to a vertical shaft making 110 revolutions per minute.

In one compartment, sufficient zinc dust is added to precipitate the solution to as near as a "trace" silver as possible, the average result being 0.04 oz. silver per ton. In the other compartment, less zinc dust is used and the precipitated solution usually assays about 0.02 oz. silver per ton. Precipitated solution is drawn through each set of 18 leaves and discharged into 24 by 7 ft. redwood tanks by a No. 3 Krogh centrifugal pump, which maintains a vacuum of about 22 in. Motors supplying power to these pumps are provided with a float-switch control, which keeps the leaves always submerged under several inches of solution, thus avoiding exposure of the precipitate to aeration and re-solution. The nearly "barren" solution, which is about one-third the total volume of solution precipitated, is used as filter press wash; and the partly precipitated solution is pumped to battery storage tanks for re-use in the stamp mortars on incoming ore.

The foregoing assay results on the precipitated solution cover a period of nine months previous to the installation of the "Crowe" process for de-oxygenating the solution prior to precipitation. During this period it was found impossible to precipitate to a "trace" silver, even when as much as 15 per cent excess zinc dust was added. For the last two months, the "Crowe" process has been in use and improved precipitation has been obtained. The "barren" solution frequently shows a "trace" of silver, with but 2.5 per cent excess zinc in the precipitate.

Clean-up of the precipitate on the filter leaves is made twice each month, the procedure occupying two men about four hours. Each leaf is raised slowly, by chain-hoist suspended from an overhead trolley, and is hosed off with solution into the precipitate compartment by a man on each side of the leaf. The bottom of the compartment is bowl-shaped, with a 4-in. pipe exit at the center of the bowl. The precipitate is pumped from the bowl into two redwood tanks, 10 ft. diameter by 3 ft. high, each equipped with a false bottom, blanket and canvas covered. Excess solution is drawn through the canvas and returned to the precipitate compartments by a small vacuum pump. The precipitate, containing 25 per cent moisture, is sent to the refinery.

The moist precipitate is fluxed with borax glass, soda ash, and silica sand, and is melted in a reverberatory furnace. This furnace is built of firebrick, inclosed in a cast-iron shell, and is fired with crude oil at 30 to 40 lb. pressure. The hearth is 4 by 6 ft. in cross-section, by 18 in. deep.

When the precipitate is entirely melted, usually about four hours after charging, the drawbar is pulled out and the molten bullion runs into molds each holding about 2,000 oz. of metal. They are shipped to the Selby smelter.

The average fineness of the bars is 966 doré, of which 9.7 parts are gold. The precipitate (dry) assays 90 to 94 per cent doré. Consumption of zinc dust was 0.644 lb. per pound doré for the six-month period preceding the use of the Crowe process, and for the last two months since using the process it has been 0.52 lb. per pound doré. An analysis of average precipitate from the partly precipitated circuit for a six-month period follows:

	Per Cent		Per Cent
Gold	0.8113	Lead	0.35
Silver	92.552	Calcium oxide	0.31
Zinc	2.50	Cadmium	0.16
Selenium	1.12	Iron	0.10
Insoluble	0.55		
Copper	0.38	Total	98.83

The selenium is probably a constituent of the ore, and, being soluble in cyanide solution and precipitable by zinc, it collects in the precipitate. Cadmium is present in small percentage in the zinc dust, and, as it is insoluble in cyanide solution, it remains in the pricipitate.